C. M. Goodale.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING

ENGINEERS.

VOL. XLVI.

CONTAINING PAPERS AND DISCUSSIONS OF THE BUTTE MEETING, AUGUST, 1918, RELATING TO THE GEOLOGY AND MINERAL RESOURCES OF MONTANA, AND THE PREVAILING PRACTICE IN MINING AND METALLURGY.

PREFACE.

This volume is the second of the three volumes which contain the Papers and Discussions presented at meetings of the Institute during the year 1918, and to it have been assigned the papers of the Butte meeting, August, 1918, which describe the geology and mineral resources of the State of Montana and the mining and metallurgical practice obtaining in the Butte district. The remaining papers of the Butte meeting and the papers of the New York meeting, October, 1918, are included in Vol. XLVII. The papers herein are classified as follows: Geology, Mining, Concentrating, Smelting, Refining, Power, and Miscellaneous. Having in view the complete exposition of the natural resources and mineral development of the State here presented, it has seemed permissible to name the book The Montana Volume.

For the magnificent series of papers bound in this volume we are indebted chiefly to the Committee on Precious and Base Metals, and especially to its Chairman, Charles W. Goodale. It is a pleasure to have an opportunity of recording here our gratitude to Mr. Goodale and his associates for this most notable contribution to the literature of the Institute.

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ERRATUM.

Page 4, line 11 from bottom. For " 6,000,000 lb. of zinc," read "120,000,000 lb. of zinc."

Proceedings of the One Hundred and Fifth Meeting, Butte, Montana, August, 1913.

COMMITTEES.

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Frederick Laist,	E. M. Dunn,	C. D. Demond,
L. V. Bender,	H. S. Ware,	H. L. Welsh.

The opening session was held on Saturday evening, Aug. 16, 1913, at the Court House, Great Falls, Mont., and was called to order by Charles W. Goodale, who, after a short address of welcome, introduced President Charles F. Rand, who responded to the address of Mr. Goodale and presided thereafter at the meeting.

The following papers were then presented in oral abstract by their authors, or authors' representatives:

Hydro-Electric Development in Montana, by Max Hebgen. Discussed by Frank Scotten, Joseph W. Richards, and another.

Notes on the Great Falls Electrolytic Plant, by W. T. Burns. Discussed by Joseph W. Richards.

The Great Falls Flue System and Chimney, by C. W. Goodale and J. H. Klepinger. Discussed by Joseph W. Richards and J. H. Klepinger.

The Determination of Arsenic and Antimony in Converter and Electrolytic Copper, by E. E. Brownson.

Great Falls Converter Practice, by A. E. Wheeler and M. W. Krejci. Discussed by Bradley Stoughton, Joseph W. Richards, S. LeFevre, and M. W. Krejci.

* The Smelting of Copper Ores in the Electric Furnace, by D. A. Lyon and R. M. Keeney. Discussed by M. W. Krejci, C. D. Woodward, Joseph W. Richards, Bradley Stoughton, and D. A. Lyon.

The following papers were read by title:

* Preparation of Ore Containing Zinc for the Recovery of Other Metals such as Silver, Gold, Copper, and Lead by the Elimination and Subsequent Recovery of the Zinc as a Chemically Pure Zinc Product, by S. E. Bretherton.

Some Recent American Progress in the Assay of Copper-Bullion, by Edward Keller.

The second session was held at the Auditorium in Butte, Mont., on Monday, Aug. 18, 1913, at 9.30 a. m. After an address of welcome by the Hon. Cornelius F. Kelley, which was responded to by President Charles F. Rand, the following papers were presented in oral abstract by their authors, or authors' representatives:

Ore Deposits at Butte, Mont., by Reno H. Sales. Discussed by L. C. Graton, W. C. Ralston, Joseph W. Richards, and R. H. Sales.

Mineral Associations at Butte, Mont., by D. C. Bard and M. H. Gidel.

Timbering in the Butte Mines, by B. H. Dunshee.

Shaft-Sinking Methods of Butte, by Norman P. Braly.

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* Held for publication in vol. xlvii.

The Use of Electricity in Mining in the Butte District, by John Gillie.

The last three papers were discussed jointly by G. E. Moulthrop, G. A. Packard, and Norman P. Braly.

The following papers were read by title:

* Notes on the Occurrence of Some of the Rarer Metals in Blister Copper, by Anton Eilers.

* Biographical Notice of John Fritz, by H. S. Drinker and R. W. Raymond.

* Assay for Gold and Silver by the Iron-Nail Method, by E. J. Hall and C. W. Drury.

* An Assay for Corundum by Mechanical Analysis, by W. Spencer Hutchinson.

The third session was held at the Auditorium in Butte, on Monday evening, Aug. 18, 1913, at 8 p. m.   Mr. Charles W. Goodale presided.   The following papers were presented in oral abstract by their authors, or authors' representatives:

Development of the Basic-Lined Converter for Copper Mattes, by E. P. Mathewson.   Discussed by Joseph W. Richards and C. D. Demond.

Roasting and Leaching Tailings at Anaconda, Mont., by Frederick Laist.   Discussed by J. C. Dick and F. Laist.

* The Use of the Microscope in Mining Engineering, by F. W. Apgar.   Discussed by L. C. Graton.

The Evolution of the Round Table for the Treatment of Metalliferous Slimes, by Theodore Simons.

The following papers were read by title:

Mining Cost Accounts of the Anaconda Copper Mining Co., by H. T. Van Ells.

The Development of Blast-Furnace Construction at the Boston & Montana Smelter, by John A. Church, Jr.

The Tooele Plant of the International Smelting & Refining Co., by N. H. Thomson and L. T. Sicka.

* Hardinge Mills vs. Chilean Mills, by Robert Franke.

* The New International Diamond Carat of 200 Milligrams, by George F. Kunz.

* The Laws of Jointing, by Blamey Stevens.

* Method of Testing Draeger Oxygen Helmets at the Copper Queen Mine, by Charles A. Mitke.

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* Held for publication in vol. xlvii.

The fourth session was held in the Margaret Theatre, at Anaconda, Mont., on Tuesday, Aug. 19, 1913, at 2.30 p. m.   An address of welcome by E. P. Mathewson was responded to on behalf of the Institute by Prof. Joseph W. Richards.   Mr. Mathewson presided at the meeting.

The following papers were presented in oral abstract by their authors, or authors' representatives:

Concentration of Slimes at Anaconda, Mont., by Ralph Hayden.

The Great Falls System of Concentration, by Albert E. Wiggin.

The Anaconda Classifier, by Robert Ammon.

Application of Hindered Settling to Hydraulic Classifiers, by E. S. Bardwell.

Determination of Gases in Smelter Flues; and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont., by Edgar M. Dunn.

Arsenic Trioxide from Flue Dust, by James O. Elton.

Ore-Dressing Improvements, by Robert H. Richards.

* Topographic Maps for the Mining Engineer, by E. G. Woodruff. Discussed by C. W. Goodale, F. A. Linforth, E. P. Mathewson, E. W. Parker, Van. H. Manning, and John Gillie.

The fifth session was held at the Auditorium, in Butte, Mont., on Wednesday, Aug. 20, 1913, at 9.30 a.m.   Prof. Joseph W. Richards presided.   The following papers were presented by their authors, or authors' representatives:

The Southern Cross Mine, Georgetown, Mont., by Paul Billingsley.

* The Reducibility of Metallic Oxides as Affected by Heat Treatment, by W. McA. Johnson.

* Rock-Drilling Economics, by William L. Saunders.

A Note on the Occurrence and Manufacture of Refractories in Montana, by W. H. Gunniss.   Discussed by Joseph W. Richards and E. P. Mathewson.

Applied Geology in the Butte Mines, by F. A. Linforth.   Discussed by B. H. Dunshee.

The following papers were presented by title:

Thermal Effect of Blast-Furnace Jackets, by Robert P. Roberts.

* The Discovery and Opening of a New Phosphate Field in the United States, by C. Colcock Jones.

The sixth session was held at the Auditorium, in Butte, Mont., on Wednesday, Aug. 20, 1913, at 8.30 p.m.   First Vice-President Benjamin B. Thayer presided.   The following papers were presented in oral abstract by their authors, or authors' representatives:

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* Held for publication in vol. xlvii.

The Precipitation of Copper from the Mine Waters of the Butte District, by John C. Febles. Discussed by Joseph W. Richards.

Notes on the Electrolytic Refining of Copper Precipitate Anodes, by W. T. Burns.

The Compressed Air System of the Anaconda Copper Mining Co., Butte, Mont., by Bruno V. Nordberg.

Increasing the Efficiency of MacDougall Roasters at the Great Falls Smelter of the Anaconda Copper Mining Co., by F. R. Corwin and S. S. Rodgers.

* The Tin Situation in Bolivia, by Howland Bancroft.

Notes on the Metallography of Refined Copper, by E. S. Bardwell.

The Coal Fields of Montana, by Eugene Stebinger.

The following papers were presented by title :

Monolithic Magnetite Linings for Basic Copper Converters, by A. E. Wheeler and M. W. Krejci.

The Electrification of the Butte, Anaconda & Pacific Railway, by R. E. Wade.

* The Substitution of Air for Water in Diamond Drilling, by Ralph Wilcox.

Lead-Silver Mines of Gilmore, Lemhi County, Idaho, by Ralph Nichols.

The Metaline Plant of the Inland Portland Cement Co., Metaline Falls, Wash., by Milo W. Krecji.

* Valuation of Coal Land, by H. M. Chance. Discussed by G. H. Ashley.

Cement Materials and the Manufacture of Portland Cement in Montana, by W. H. Andrews.

Excursions and Entertainments.

The members and guests assembled at the Rainbow Hotel, Great Falls, Mont., on the morning of Saturday, Aug. 16, where registration facilities, badges, booklets, and programs were provided, and where they were met by the Local Committees.

The members were then taken by automobile to visit the hydro-electric developments at Great Falls, and Rainbow Falls, on the Missouri river, with a visit to the Giant Springs on the return trip.

In the afternoon the party was again taken by automobile to the plant of the Boston and Montana Smelter, every department of which was offered for inspection, and explained by members of the metallurgical staff.

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* Held for publication in vol. xlvii.

On Sunday, Aug. 17, the members left Great Falls for Helena, where luncheon was served at the Placer Hotel through the hospitality of the Local Committee, after which a sight-seeing trip about the city was made by some of the members and guests, while others visited the lead-silver smeltery of the American Smelting & Refining Co.

At 4.20 p.m. the party left Helena, arriving in Butte in the evening. The headquarters of the Institute at Butte was at the Silver Bow Club, which Club, together with the University Club and the Butte Country Club, extended its courtesies to the visiting members and guests during their stay.

On the morning of Monday, Aug. 18, the ladies of the party were entertained by the wives of the local Institute members on a sight-seeing trip about the city of Butte, with a luncheon at the Silver Bow Club, and a visit in the evening to the Butte Country Club as the guests of Mrs. Dr. F. W. McCrimmon.

In the afternoon competent guides were provided for visits by members and guests to any one of the following points of interest:

Inspection of electrically-driven air-compressor plants and air hoist at the High Ore mine.

Inspection of the underground workings and pumps at either the Original, Leonard, North Butte, or Tramway mines.

Inspection of the underground workings and of the zinc mill of the Butte & Superior Copper Co.

Inspection of the mine and of the copper-leaching plants of the Bullwhacker Copper Co., and of the Butte & Duluth Mining Co.

General surface trip of inspection of the mines and surface geology of the Butte district.

On Tuesday morning the party was taken by special train to the Washoe Smeltery, at Anaconda, where they were conducted by the Local Committee on a trip of inspection through the plant. A special train then took the party to the town of Anaconda, where luncheon was served at the Montana Hotel.

In the afternoon, after the technical session, the special train carried the members and guests back to Butte.

On Tuesday evening the party assembled at the Auditorium, where some extraordinarily interesting moving pictures by Max Hebgen, illustrating the hydro-electric development of Montana, were presented. Following this special cars carried the members and guests to Columbia Gardens, where a very pleasant social evening was spent visiting the gardens, with dancing by those who cared to.

On Wednesday the ladies were carried by automobile to the Basin Reservoir as guests of the Butte Water Co. Luncheon was served,

Mrs. Eugene Carroll acting as hostess, assisted by the wives of the local members. The afternoon was spent in boating on the lake, and other social recreations.

In the afternoon the men of the party made visits of inspection to the different points of technical interest in Butte, listed above.

On Thursday, Aug. 21, a special train conveyed the party to the Southern Cross mine, where an exceedingly pleasant day was spent socially and in visiting the mine. A very good luncheon was served at the mine from dinner buckets.

The meeting closed with a banquet, which was given at the Silver Bow Club at 8.30 p.m. Mr. William L. Saunders acted as Toastmaster, and the following toasts were responded to:

Cornelius F. Kelley, "Butte."

Charles F. Rand, "The A. I. M. E."

Lee Mantle, "Montana."

J. H. Durston, "The Press."

Bradley Stoughton, "Co-operation for Institute Progress."

One hundred and sixty-two members and more than 300 guests attended the meeting. The attendance at technical sessions varied from 125 to 182 each.

PAPERS.

Ore Deposits at Butte, Mont.

BY RENO H. SALES, BUTTE, MONT.

TABLE OF CONTENTS.

INTRODUCTION.

THE geology of Butte possesses especial interest on account of the magnitude of the ore deposits, their extraordinary richness and persistence in depth. Since its discovery in the early 60's the district has yielded, in round numbers, 6,000,000,000 lb. of copper, 260,000,000 oz. of silver, 1,250,000 oz. of gold, more than 6,000,000 lb. of zinc. This enormous metal production has been derived from an ore tonnage estimated roughly at 65,000,000 tons. At the present day, most of the important mine shafts have reached a depth of 2,400 ft., while several are more than 3,000 ft. deep. At these great depths rich bodies of copper ore are being exploited, showing no diminution in metal content as compared with the ores of the higher levels.

The present (1913) daily rate of production is approximately 14,000 tons, of which 13,250 tons may be regarded as copper ore, although a small percentage of this is minable only,because of the presence of

notable amounts of silver, and the remaining 750 tons are zinc ores. In addition, a small tonnage of straight silver ores is mined, chiefly by leasers, from abandoned silver mines. A small amount of copper is recovered in the precipitation plants treating the copper-bearing waters pumped from the mines.

In spite of their great importance as metal producers, the subject of the geology of these ore deposits received but scant attention at the hands of investigators until recent years. The Butte Special Folio,[1] the work of S. F. Emmons, W. H. Weed, and G. W. Tower, published by the U. S. Geological Survey in 1897, was the first publication to deal with the Butte district in detail. This was followed by a more comprehensive study undertaken by W. H. Weed[2] during the period extending from 1901 to 1905, the results of his work appearing in 1912. Other articles treating special features of the ore deposits have appeared from time to time. H. V. Winchell's paper[3] dealing with the artificial production of chalcocite or copper glance was timely and of great interest in connection with the problem of chalcocite formation. More recently Charles T. Kirk[4] has carefully studied the various phases of granite alteration found in association with the copper veins. He endeavored to determine if any genetic relation existed between these alteration phases and the deposition of certain copper minerals. He concludes from his investigations that the chalcocite of the Butte veins is almost entirely a product of descending sulphide enrichment, a view expressed also by W. H. Weed.

Weed's study was concluded in 1905 and since that time much development work has been done. New mines have been opened and the older ones have been greatly extended both vertically and laterally, bringing to light many new and interesting features concerning the vein and fault structure. The persistence of rich ore bodies to great depth has aroused a keen interest among geologists regarding the manner of formation of the abundant copper mineral chalcocite. Much information is available bearing directly upon this important question. In the presentation of this new material, together with such associated facts as are considered to be necessary and of general interest concerning the geology of these ore deposits, the writer hopes to find justification for the publication of this paper.

[1] *Butte Special Folio* (No. 38), *Geologic Atlas, U. S. Geological Survey* (1897).

[2] Weed, W. H., *Professional Paper No. 74, U. S. Geological Survey* (1912).

[3] Winchell, H. V., Synthesis of Chalcocite and Its Genesis at Butte, Montana. *Bulletin of the Geological Society of America*, vol. xiv., pp. 272 to 275 (1902).

[4] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (Jan., 1912).

GENERAL GEOLOGY.

Butte is situated near the western border of a large granite area, called the "Boulder" batholith, extending southwesterly from Helena to the Big Hole river in Beaverhead county, a distance of 70 miles. The general outline of the batholith is oblong, but it is extremely irregular in width, averaging about 20 miles. Many smaller isolated masses of similar rock occur along its western and northern borders. The granites of Marysville, Philipsburg, Clinton, Garnet, and several other mining districts are undoubtedly offshoots from a main parent magma. It is not improbable that further study will show a close relation between the Boulder granite area and the great granite batholith of central Idaho.

The Boulder batholith made its appearance subsequent to the great Rocky Mountain building period, marking the close of Cretaceous times. It is more recent than the large mass of andesite forming Bull mountain, near Whitehall, which is known to intrude upturned Palæozoic and Cretaceous rocks. There is an abundance of evidence to show that the Boulder batholith did not produce a doming effect on the sedimentary rocks now found along its borders; in fact, the orientation of these intruded rocks does not appear to have been visibly disturbed even where they are in direct contact with the granite. In nearly every instance where such contacts are open to observation, the sedimentaries are found to dip at a steep angle toward the granite; an exception, however, may be noted at Elkhorn, where dips away from the granite appear to be a coincidence and not an effect produced by its intrusion. In a broad way the batholith seems to occupy the trough of a great synclinal basin in whose dissected sloping sides may be seen remnants of the entire series of sedimentary rocks reaching from the pre-Cambrian slates and shales upward to the coal-bearing sandstones of the late Cretaceous. The manner in which this great displaced mass of sedimentaries made its escape is not known.

While the chemical composition of the granite is fairly uniform over the whole area there is a slight variation in physical character or texture, indicating the possibility of two or more distinct intrusions. The texture is usually granitic, in fact, typically so, but at times it is decidedly porphyritic, showing numerous imperfectly formed orthoclase feldspars an inch or more in length. Basic segregations composed of dark silicates are present in considerable amounts as a noticeable feature of the marginal areas of the batholith. These small dark masses, which are from 1 to 6 in. in diameter, are possibly more prevalent where the granite has the even texture and the porphyritic character is lacking. They are more resistant to weathering than the body

of the granite, so that they often stand out in relief over the surfaces of the big rounded boulders, giving a "warty" appearance to the rock.

These variations in texture may be seen in the mines of the Butte district, but there is little, if any, direct evidence to support the hypothesis of two or more separate and distinct intrusions. It is more probable that these variations in texture were developed during the process of cooling and were due to uneven temperature conditions within one great granite area.

An analysis of the typical Boulder granite is given by Weed as follows:

SiO_2	67.12
Al_2O_3	15.00
Fe_2O_3	1.62
FeO	2.23
CaO	3.43
MgO	1.74
K_2O	4.52
Na_2O	2.76
TiO_2	0.48
MnO	0.06
BaO	0.07
SrO	0.03
P_2O_5	0.15
H_2O below 110°,	0.09
H_2O above 110°,	0.58
	99.98

The cooling of the main granite mass was accompanied in later stages by the segregation of great quantities of aplite, in the form of dikes or often as irregular masses of all sizes from a few yards to a mile or more across. These aplite areas are especially abundant near Butte, and also west and northwest of Boulder in Jefferson county. The texture of the aplite is usually fine grained, though often grading into a coarse pegmatitic structure or, in extreme stages, showing a development of pure glassy white quartz. The formation of aplite seems to have taken place at a very early period; its presence, therefore, does not influence in any manner the occurrence of ore deposits throughout the Boulder granite area.

A large body of dacite-rhyolite, partly intrusive and partly extrusive, covers a goodly portion of the western half of the Boulder batholith. Many smaller isolated plugs and dikes of rhyolite occur in various parts of the granite area, many of which are associated with ore deposits. A notable instance is at Butte, where the ore formation, however, is much older than the rhyolite. These rhyolites, which at one time probably covered a large portion of the granite

batholith, were intruded into the granite long after the latter had entirely cooled and at a time when the present-day physiographic features were strongly developed.

Rocks of the Butte District.

Granite.

Granite forms the main body of rock inclosing the Butte ore deposits. It is properly a quartz-monzonite, but the term "granite," or "Butte granite," has been so universally employed that it seems advisable to adhere to common usage in this paper. The Butte granite, which is in fact a local area of the Boulder batholith, is a dark gray-green rock of medium coarse granitic texture, but exhibiting at times a more porphyritic phase. This latter variation is especially prevalent in the Steward, Gagnon, and Colorado mines in the southwestern part of the district. The granite everywhere exhibits well-defined systems of joint planes, which, however, do not bear any definite relation to the later fracture systems. The jointings are old, and in many instances in the Butte mines show evidence of slight faulting movements along them, a condition brought about by faults or readjustments taking place within broad inter-fault blocks. While the joint planes have played no part in the determination of the larger features of the fracture systems, they have been of the greatest importance in making the granite adjacent to the fissures more permeable to ore-bearing solutions, and the existence of many of the large "stock-work" ore bodies of the Leonard mine was made possible through the presence of joint planes.

In the veins and intervening country rock in the copper vein area the granite has suffered intense alteration, so that often its original character is only in part preserved.

Aplite.

Small irregular bodies and dikes of aplite are abundant in the northwestern portion of the Butte district. This rock has been noted in unusually large amounts in the Green Mountain, Mountain Con, and Corra mines. In the main copper belt of Anaconda hill aplite is rarely seen. Immediately to the west of Big Butte it covers an area of several square miles. This exceptionally large body is separated in a general way from the copper zone by the intrusive rhyolite plug forming Big Butte. Where aplite occurs in the form of dikes, which may vary in thickness from a few inches to 50 ft. or more, there is no uniformity in dip or strike, although they generally lie at a low angle. The texture of the aplite is usually fine grained,

though pegmatitic phases are common. In general, it is not often seen as a wall rock of the ore bodies, for, by volume, it forms but a small percentage of the whole rock mass in which the veins occur.

Quartz-Porphyry.

In the copper belt of Butte the granite has been intruded by a series of roughly parallel quartz-porphyry dikes which extend in a general easterly and westerly direction across the district. These dikes are relatively narrow, from 10 to 50 ft. wide, but persistent, and they follow closely the general trend of the earliest system of copper veins, thus indicating a close genetic relation between the oldest veins and the porphyry. The dikes, however, are prior to any known vein formation and apparently intruded the granite before the latter had entirely cooled. The vein and fault fracture systems intersect the granite, aplite, and quartz-porphyry dikes alike, and all these rocks are similarly affected by the general alteration processes accompanying later vein formation.

Rhyolite.

Rhyolite, both intrusive and extrusive, occurs to the west and northwest of the district. Big Butte, a prominent topographic feature, is formed of intrusive rhyolite, as is the round-topped hill a mile to the south. From these two main bodies there are many offshoots and dikes having a general north and south course, some of which are known to break through silver and copper veins, thus fixing the period of the rhyolite eruptions at a much later date than that of the quartz-porphyry. Extrusive rhyolite extends northwesterly from Big Butte, covering a large area, and in itself is probably the remnant of a much larger rhyolite flow which formerly covered a large portion of the western half of the Boulder granite batholith.

Andesite.

Isolated areas of a porphyritic andesite occur to the northwest of the district, the age relations of which have not been definitely determined. They are believed to be comparatively old and probably prior to the granite, corresponding, therefore, to the early andesites of Bull mountain north of Whitehall, Mont., which are known to be intruded by the Butte granite.

Sedimentary Rocks.

The nearest sedimentary rocks are found in the Highland mountains, 15 miles south of Butte. Here Algonkian rocks, together with Palæozoic quartzites and limestones, are intruded by the Butte granite, which in turn intrudes also earlier andesite and a basic diorite.

Lake Beds.

Three miles west of Butte is found the eastern limit of an extensive area of a geological formation termed Bozeman Lake beds by the U. S. Geological Survey. This formation is made up largely of sand, gravel, and water-bedded volcanic ash, and at some former geologic period it covered large valley areas in south central Montana. Near Butte these beds fill a long, narrow, pre-existing drainage valley, reaching from the town of Rocker to Melrose, a distance of 30 miles. A 700-ft. mine shaft sunk a half a mile northwest of Rocker discloses a depth of more than 1,000 ft. of these beds, with the total thickness still undetermined.

Valley Débris.

The bottom of the level valley to the south of Butte is composed of coarse detrital sand formed from rapidly disintegrated granite. The greatest depth of this sandy material is unknown. Shafts sunk in the valley floor east of Meaderville have shown the former erosional surface to be buried to a depth of from 200 to 400 ft. in that section, or more than 200 ft. lower in elevation than the present bed of Silver Bow creek where it leaves the valley near the site of the Colorado Smelter.

The manner of occurrence of the Bozeman Lake bed formation near Rocker and the presence of the débris-filled valley south of Butte show conclusively that the former drainage level of the Butte district has been considerably disturbed, and it is probable that the present conditions were brought about either by faulting, volcanic flow coverings, or important crustal movements. These apparent drainage level oscillations are believed by some writers to have played a tremendously important part in the formation of the bonanza chalcocite ore bodies of the Butte district. This feature will be more fully discussed in connection with the problem of ore formation.

GENERAL GEOLOGIC STRUCTURE.

Regional Faulting.

It is difficult to recognize lines of faulting in the area of the Boulder granite batholith on account of the uniformity of the rock and the frequent masking of structural features by surface débris or late volcanic flow coverings. Numerous faults are known, many of which are mineralized, but the necessary evidence is lacking to fix definitely the direction of throw and amount of displacement along these fissures. Unless dikes or veins are intersected, no clue is afforded as to their magnitude other than the width, amount of fault clay, crushed wall rock, etc.

Taking a broad view of the Boulder batholith, it is of interest to note that fissuring is a common feature over the whole area. While no mapping has been done which permits of an accurate classification of these fissures, it is known that in general they may be grouped into two main series: (a) those having a general east-west strike, and (b) those having a northwest-southeast strike; both groups corresponding in a broad way with the two most important fracture systems of the Butte district. In general, however, these fissures are more or less localized, and it is not proper at this time to regard them as planes of movement resulting from great regional disturbances.

Of the recognizable faults of this class the only one known to be directly concerned with the geology of the Butte ore deposits is the Continental fault, lying in a north and south direction along the base of East ridge near the Pittsmont mine. There is but little doubt that this fault is of comparatively recent origin, and that it has had an important influence in determining the present topography of the district. The dip is to the west at a steep angle, Butte being on the downthrown, or hanging-wall side. The amount of vertical displacement is probably in the neighborhood of 1,500 ft., while the direction and amount of horizontal displacement have not been determined. A more detailed description of this important fault will be found on a later page. (See pp. 29–30.)

STRUCTURE OF THE BUTTE DISTRICT.

Locally, the rocks of the district are intersected by many separate and distinct periods of fissuring. These fractures cut alike the granite, aplite, and quartz-porphyry, but the rhyolite plug forming Big Butte has been found in many instances to be later, often cutting through veins belonging to certain fissure systems. These observed facts throw considerable light on the relations between the aplite, quartz-porphyry and the later rhyolite eruptions, but owing to lack of mine development in the western part of the district it is, as yet, impossible to correlate definitely the rhyolite intrusion with any particular period of fracturing or mineralization in the copper or silver veins. The rhyolite is known to be younger than the earliest vein system, but further than this the exact time of the eruption is not known.

Plate I. has been prepared to illustrate the structural relations of the various fissure systems. It is a horizontal section taken approximately at an elevation of 4,600 ft. above sea level, corresponding to an average depth, in the mines, of 1,500 ft. below the surface. Owing to the small scale of the map and the complexity of the vein and fault structure it has been possible to represent only the larger

and more general features. The Anaconda fractures, which are continuously mineralized, are denoted by solid black. The vein filling other than the crushed granite, in the later fissures is also represented by solid black whether it be of commercial grade or not. Indeed, in many instances, the indicated vein filling is not of commercial grade. The structural features are well developed by mine openings, and therefore the map may be regarded as only slightly idealized.

Plate II., a north and south vertical section through the copper district in the vicinity of the Anaconda shaft, indicates the structural relations of intersected veins and fissures on dip. Since the plane of section meets the Blue and Rarus faults at acute angles the dips indicated are flatter than the true dips.

Classification of Fissures.

Grouped according to their relative ages the fissures of the Butte district may be divided into six distinct systems, as follows:

1. Anaconda or east-west system, comprising the oldest known fractures.
2. Blue system, the earliest fault fissure.
3. Mountain View breccia faults.
4. Steward system.
5. Rarus fault.
6. Middle faults.
7. Considered as a local feature the Continental faulting may be regarded as a seventh separate period of fissuring.

This classification is based on information gained through a close study of the intersections of the various fissures, aided to some extent by strike and dip and their mineralogical and physical characters. While the development by actual mine openings generally determines definitely the relative ages of intersecting veins or fissures, it is often desirable to learn in advance, if possible, the age relations of certain veins or fractures before such intersections are reached by mine workings. In these cases the vein characteristics must be relied upon for proper guidance. As a general rule the mineralogical composition offers but slight assistance, being a factor depending on geographical location rather than on relative geologic age. Where intersections are not available, the physical character of the fissure together with the determined strike and dip is of importance. As will later appear, the Anaconda fractures are more solidly and continuously mineralized and have a more nearly east and west strike than the later systems. The Blue system fissures are typical fault fissures, with ore occurring in disconnected shoots, and they have a remarkably uniform strike of about

N. 60° W.　The Steward fissures are characteristic faults, with a slightly north of east strike, and contain ore more sparingly than the Blue fissures.　The Rarus and later fractures are typical fault fissures of marked movement and contain no ore other than fragments or blocks dragged from older veins.

When a typical fault fissure is penetrated by a mine opening and is found to carry no vein mineral or ore where encountered, the strike and dip are the factors used in provisionally assigning it to a recognized fissure system.

The Mountain View breccia faults have certain physical characteristics by which they are readily distinguished from all other fracture or vein systems.

Difficulties in classification are met with in certain areas, where the Anaconda fractures are but slightly mineralized and some secondary movement has taken place along them, resulting in a vein which closely resembles the later faults.　Again, in the Mountain View, West Colusa, and Leonard mines, many of the Blue fissures are solidly mineralized over hundreds of feet, making them identical in physical appearance with the older veins.　A further complication arises in this section owing to the presence of many ore-bearing cross fractures, which, though parallel to the Blue fissures, are in fact older and belong to the Anaconda system.

A more general classification or grouping of these fissures may be made which is to some extent a genetic one, but largely one having to do with the physical nature of the fissures.　This reclassification or grouping is offered here in an endeavor to set forth more clearly to the reader the physical differences between the earliest-formed veins and those of later age known as fault veins.　A clear understanding of these differences will be of material aid in forming a proper conception of the main structural features of the faults and ore deposits, which at times are bewildering in their complexity.

From a study of the occurrence and nature of the various fissures of the district it is found that they may be arranged in three general groups, as follows:

Group A.—Remarkably continuous complex fractures or fissures of but slight displacement, with but little or no crushing of the wall rock; varying greatly in strike and dip, and exhibiting at times a tendency to develop highly fissured areas in which there may be found a multiplicity of transverse fractures more or less at right angles to the general direction or strike of the main fracture planes.　The whole Anaconda fracture system falls within this group.　As mineral veins they are in general uniformly and continuously mineralized.

Group B.—Persistent well-defined fissures of marked displacement and of later age than Group A. The fissures of this group are typically fissures of faulting, being invariably accompanied by crushed granite, attrition clay or gouge, etc. It is seldom that these evidences of movement are entirely obscured by later mineralization. In marked contrast to the fractures of Group A, these fissures are not continuously mineralized, the mineral bodies occurring in disconnected shoots. Included in Group B are the ore-bearing faults of the Blue and Steward systems, also the later unmineralized fissures such as the Rarus, Bell, Middle, and Continental.

Group C.—Breccia-filled angular cracks or fractures later in age than Group A, but prior to some of the faults of Group B. These angular cracks exhibit no displacement. They appear to have been formed by the simple drawing apart of the wall rocks. The Mountain View breccia faults and the ore breccias of the Gagnon mine are the important examples of this group.

Anaconda Fracture System.

The Anaconda system includes the earliest-formed fractures of the district. It embraces, therefore, all fissures known to be earlier than the oldest known faults, which are those comprising the Blue system. It is not absolutely certain that all of the old fractures provisionally placed in the Anaconda system are exactly of the same age, but they are practically so. Such additional, recognizable fractures which are older than the Blue fissures and yet are later than the oldest known mineralization are of minor importance and not resolvable into a distinct system. Where such early fissuring is observed, cutting older fractures, it is believed to have resulted from continued movement or slight readjustments along the main fracture lines during the active period of mineralization.

The Anaconda fissures are the oldest geologically and the most important commercially in the district. This group has formerly been called the "East-West" or "Quartz-Pyrite" system of veins, both misnomers, strictly speaking, because the fractures exhibit wide variations from an east-west strike, and the mineral content of the Butte veins is not necessarily an incident of geologic age, but rather one of geographic position. This system is represented by a series of complex fractures, of a general east-west strike, extending across the district. In a broad way, in the copper-producing area they may be divided into two groups, one north of the other and separated from the latter by a relatively barren area. The northerly group embraces the Syndicate, Bell-Speculator and nearby fractures, and

the southerly comprises the Anaconda, Moonlight, O'Neill, and others.

North Group.—The most important fracture zone of this group is the Syndicate vein. In adjoining properties it is successively called the Yellow Jacket, Poulin, Buffalo, Mountain Con, Wake Up Jim, Middle vein, Bell-Speculator vein, etc. Westerly from the Mountain Con and Green Mountain mines the Syndicate vein is usually a well-defined, single, mineralized fissure, rarely branching excepting for short distances, forming "horses," or included barren granite blocks. Toward the east, however, it divides, and its various branches take a southeasterly course toward the Mountain View mine and split further into smaller veins until the identity of the major fissure is completely lost. These smaller branches form in a general way, no doubt, a connecting link with the great Anaconda complex in the vicinity of the Mountain View mine.

Other important fissures of the north section of the copper-producing district belonging to the Anaconda system are the Badger State, Berlin, North Wild Bill, Modoc, Mountain Con South, Eastwest Gray Rock, and the North vein of the Mountain Con mine. In addition to those above named there are numberless small cross fissures and cracks, associated with the larger fissures, and of geological interest, but not important as ore producers.

The main Syndicate vein in the Mountain Con and Buffalo dips 88° to the south. It is variable, however, at times dipping slightly north. Southeasterly it has a southerly dip of 60°. The North Wild Bill is vertical or slightly south dipping. The Badger State dips north at the surface, changing to a steep south dip in depth. The Eastwest Gray Rock and Modoc fissures dip 65° south, and the North vein of the Mountain Con dips 50° to the south.

South Group.—The south fracture complex known as the Anaconda vein in the Never Sweat, Anaconda, and St. Lawrence mines is of vastly more commercial importance than the north group. Beginning on the extreme west in the Gagnon mine, where it forms a compound fissure zone combined with later faulting, and passing easterly, it is known successively as the Gagnon, Original, Parrot, Anaconda, and Mountain View South vein, etc. Although intersected and displaced many times by later faults the identity of the main fissure is not lost. Easterly from the St. Lawrence mine this fracture exhibits great complexity. Passing through the Mountain View mine there is developed a great fissured zone bounded on the north by the Shannon vein, the easterly extension of which is the Colusa vein, and on the south by the Mountain View South No. 2 vein, the last named being
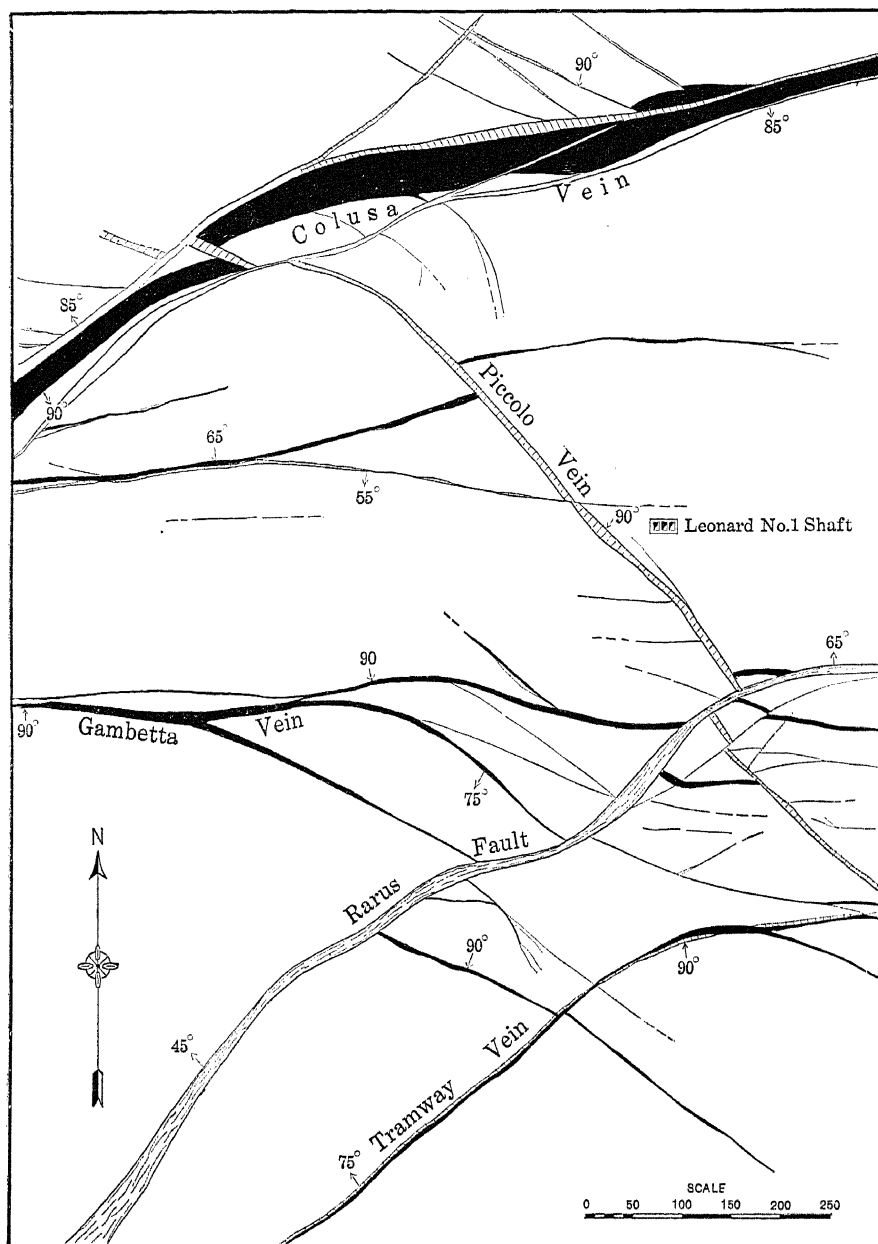
FIG. 1.—PLAN OF A PORTION OF THE 300-FT. LEVEL OF LEONARD MINE, SHOWING
VEIN STRUCTURE.

the faulted extremity of the main Anaconda fissure. The Shannon or north bounding fissure has a strike of N. 75° E., and a dip of 85° north, while the South No. 2 vein strikes S. 75° E., and dips 65° south. In its southeasterly course the latter branches into many fissures, forming many smaller veins of the Rarus and Berkeley mines; the identity of the principal fissure, however, is lost.

The distance between the limiting Shannon and South No. 2 fissures at the Mountain View shaft is 400 ft. For several hundred feet east of the Mountain View shaft there are a great number of connecting cross fractures between the Shannon and South No. 2 fissure. These cross fractures have a general strike of N. 20° W., with a westerly dip of about 75°. In many instances these cross veins have proved to be good producers of ore, although they are not of great width. The structure here is much complicated by the presence of several mineralized fissures belonging to the Blue system, being in fact the southerly extensions of the Gray Rock fault veins.

Owing to the divergence on strike of the Shannon and South No. 2 veins, the space between these limiting fissures becomes greater toward the east. There is an increasing tendency on the part of the Shannon vein to throw off north-south fissures in a southerly direction. In the West Colusa upper levels enormous stopes were made, using the north boundary of the Shannon fissure as the north wall, while the larger portion of the material mined out was mineralized granite intersected by a multiplicity of closely spaced fissures having a general strike of about N. 20° W., or nearly at right angles to the strike of the Shannon fissure. In the Leonard mine the north and south fissuring is not present in the upper levels. (See Fig. 1.) The first tendency toward a development of transverse fracturing appeared on the 600-ft. level. At greater depths the north-south fissuring became more intensified, while the east-west fissures became gradually less prominent, until it was found that below the 1,200-ft. level only the great areas of closely spaced north-south fissures remained, bounded on the north by the limiting north wall of the Shannon fissure, and forming the great " horse-tail " ore bodies of the Leonard, Tramway, and West Colusa mines. (See Fig. 2.)

Going northeasterly, the termination of the Shannon-Colusa-Leonard vein is remarkably abrupt and gives rise to a most peculiar geological condition. The throwing-off tendency of the Shannon fissure reaches a climax near the Leonard shaft, where the east-west fissuring ceases and the identity of the Shannon-Colusa vein is completely lost in a perfect maze of north and south fissuring without definite boundaries. (See Plate I.)

Lying to the south and parallel to the main Anaconda fissure are two important veins known as the Moonlight and O'Neill veins. The O'Neill fissure is distant about 1,000 ft. from the Anaconda, and it forms the general southern boundary of the great zone of alteration. The eastern extension of the O'Neill vein is known as the Pennsylvania No. 1 vein in the Pennsylvania mine, and as the Silver Bow vein farther to the southeast. The Moonlight fissure is prominent in the Moonlight and Anaconda mines, but dies out in an easterly direction before reaching the Pennsylvania shaft.

Many small unimportant fractures belonging to the Anaconda series occur south of the O'Neill vein, notably the Cambers, J. I. C., Glengarry, Silver Bow No. 3, and others, but these have been only slightly developed, owing to their non-productiveness in depths greater than 400 ft.

Southeasterly from the Rarus, Berkeley, and Silver Bow mines, fissures of the Anaconda system pass into the property of the East Butte Mining Co., following in a general way the course of the quartz-porphyry dikes. These relations may be readily understood by referring to Plate I.

The principal veins of the so-called silver area are believed to be mineralized fractures of Anaconda age. Owing to the mining inactivity for many years past in this area, and through the general lack of definite information concerning the geological conditions, it is not possible to map with any degree of accuracy the vein structure in that section.

Blue System of Fissures.

In the Blue system of fissures is included a series of northwest-southeast fault fissures which cut and displace the fractures of the Anaconda system, but are themselves older geologically than the Steward faults. Their general strike is in the neighborhood of N. 55° W., with extreme variations within a range of N. 30° W. to N. 75° W. There is a notable uniformity in the arrangement or spacing of the important members of this series of faults in a northwest-southeast direction, or at right angles to the general line of strike. This feature is well illustrated in Plate I.

The important fissures of the Blue system, named in order of their occurrence, beginning on the southwest, are the No. 2, No. 1, Clear Grit, Blue, Diamond South or Dernier, High Ore, South Bell, Skyrme, Edith May, Jessie, Gem, and Crœsus, all of which are mineralized and most of which contain ore bodies of immense value.

In contrast to the marked uniformity in strike, permitting accurate projections over hundreds or even thousands of feet, the dip of the

Blue veins tends toward the opposite extreme. The fissures as a rule are steep, but the variations in dip along a single fissure may cover a wide range. It is not uncommon to note a change from a north dip to a south dip, followed by a change again to the north, all taking place within a vertical range of 1,600 ft. The Edith May vein, for example, dips 80° north from the surface to the 700-ft. level; it is vertical from the 700-ft. to the 1,600-ft. level, and has a south dip below the 1,600. The Jessie exhibits tendencies even more erratic. In the Modoc mine it is very steep or slightly north, dipping near the surface, changes to a 60° south dip at a depth of about 500 ft., retaining this flatter dip to the 700-ft. level or thereabouts, where it again assumes a vertical dip to the deep levels.

While the general tendency of the fissures of this system is to dip south, some of them do not follow this rule. The most southerly ones, the Clear Grit, Flat, and Blue, have decided south dips, from 45° to 65°. Those intermediate, the Skyrme and Edith May, are more nearly vertical, while on the extreme north the tendency is to dip steeply north in the upper levels, with a change to the south at greater depths. The general variations in dip are well illustrated in Plate II. The Skyrme vein is of a particularly wavy habit on its descent into the earth.

The fissures of the Blue system are faults of marked displacement. Owing to the uniformity of the country rock in which the fissures occur it is difficult to determine accurately the direction of movement and amount of displacement. Where, however, a fault plane intersects two or more veins having different strikes and dips, these factors may be closely approximated. The amount of movement along the more important Blue fissures ranges from 150 to 300 ft. The High Ore fault vein displacement has been determined by Paul Billingsley to be in the neighborhood of 270 ft., with the line of direction of movement making an angle of 35° with the horizontal. In all of these fissures the south or hanging wall has apparently moved to the southeast and downward relatively to the foot-wall and the direction of movement of the hanging-wall has been along a line making an angle of 15° to 35° with the horizontal. The Blue faults more properly fall under the head of thrust faults, rather than normal faults as the latter term is commonly understood.

In cases where but one vein is cut by the fault plane there is a certain feature to be observed which offers a clue to the direction of movement. Large grooves, or striations, may be seen in the hard ore or the wall rock adjacent to the principal planes of movement. These grooves, or waves, measure from a few inches to a foot or

more from crest to crest, with depths of an inch or more. The writer is led to believe that they are good indications of movement because they exhibit a remarkable uniformity in dip or pitch along the walls where recognized in widely separated members of the series, and they correspond in this respect to the larger fault vein structure earlier described. This belief is corroborated by the determination in the case of the High Ore fissure above noted. The small scratches and parallel lines so commonly seen in the soft fault clay are of no value whatever as indications of direction of important movements. In a single slab of clay an inch thick striations or lines may be found extending in every conceivable direction.

Viewed in a broad way the fissures of this system show a strong tendency to branch toward the southeast, a feature well marked in the Blue, Skyrme, Edith May, and Jessie veins. Toward the west and northwest there is a noticeable swerving to a more westerly strike and many instances of unions on strike of important fissures are noted. A good example of this condition occurs in the Corra mine where the High Ore, South Bell, and Skyrme veins unite, forming a single fissure. (See Plate I.) This evident tendency to unite westerly on strike with a possible greater displacement along the large single fissure may have some significance when taken in connection with their origin.

Mountain View Breccia Faults.

A curious and interesting feature in connection with the copper veins is the occurrence of peculiar breccia faults (filled cracks) in certain localities within the copper-producing area. They are of frequent occurrence in the Mountain View and Leonard mines in the eastern part of the district and in the Gagnon mine on the west.

These breccia faults, or veins, are persistent angular fissures filled with fragmental material composed of country rock or with some fragments of earliest veins. The size of the fragments ranges from pieces a foot or more in diameter down to fine sand. Generally the breccia is a mixture of all sizes, although at times it is all fine material so arranged in the fissures that it resembles stratified sandstone. As a rule it has the general appearance of ordinary concrete, showing angular rock fragments loosely set in a matrix of finer disintegrated granite grains. Often the larger pieces are rounded and look water-worn like stream pebbles. At or near the contact or intersection with older veins the breccia frequently contains many fragments of the vein filling, in sufficient quantities at times to constitute ore. This feature is especially important in the Original and Gagnon

mines, where big stopes have been made in ore breccia associated with older veins. In fact, from the surface to the deepest levels this has been a characteristic feature of the Gagnon-Original vein.

The general strike of the Mountain View breccia faults is about N. 75° E. They extend from the most westerly workings of the Gagnon easterly through the Original and Steward mines, thence northeasterly toward the Mountain View mine. At the present time there is not sufficient mine development to determine whether a general continuity exists between the breccia faults of the Mountain View mine and those of the Original-Steward mines. In the St. Lawrence mine they are found in abundance extending northeasterly into and through the Mountain View and West Colusa mines; most of them, however, die out in the vicinity of Leonard No. 1 shaft. While the general northeasterly course is well defined, locally and in greater detail, the strike is extremely irregular. They are angular and zigzag in plan, resembling streaks of lightning. A fissure from 12 to 18 in. in width may be found regular in course for several yards, when it will suddenly offset at right angles, following a joint plane or vein wall for a distance of several feet, after which it again resumes a normal course. These freakish tendencies are well illustrated in the general vein map, Plate I.

The breccia faults vary greatly in width, even within short distances along their strike and dip. Widths of from 6 in. to 2 ft. are common in the Mountain View mine, while thicknesses of from 10 to 30 ft. are prevalent in the Gagnon mine. The north cross-cut of the 1,900-ft. level of the St. Lawrence mine develops a body of breccia 135 ft. in width. A cross-cut on the 1,800-ft. level of the Mountain View mine has been extended 80 ft. into breccia, disclosing but one wall.

Age of the Breccia Faults.—The formation of these interesting faults undoubtedly followed the period of Blue vein formation. Without exception the breccia is found to cut through the ore of both the Anaconda and the Blue vein systems. Instances have been noted where secondary unimportant movements have taken place along the Blue vein fissures, intersecting and displacing slightly the Mountain View breccia. The Rarus fault cuts and displaces the faults, but the relation between the Steward faults and the breccia has not been satisfactorily ascertained. In the Gagnon mine the breccia is often much squeezed and faulted by northeast fault movements apparently of Steward age. On the whole the evidence indicates that these breccia faults were formed subsequent to the Blue fissures and later than the ore filling in the Blue vein fissures. They were formed, however, prior to the Rarus faulting and probably slightly antedate

the Steward faults. There are indications that the breccia veins were not all formed at the same time, but that in certain instances in the Mountain View some were formed as late as the Middle faults.

The Steward Fissure System.

In the Steward system is included a series of northeast-southwest fault fissures extending across the district. They cut and displace the quartz-porphyry dikes and the veins belonging to the Anaconda and Blue systems. The Steward fissures strike slightly more north-easterly than the veins of the Anaconda system, though the angle of intersection is very acute, often forming strike faults along them. Referring to Plate I., it will be noted that these fissures are more or less regularly spaced from north to south and that the strike does not vary much from a N. 65° E. course. The dip is uniformly to the south, ranging from 50° to 75°, with an average approximating 65°.

The most prominent members of this series, naming them in order from south to north, are, the Rob Roy, No. 16, Mollie Murphy, No. 6, Steward, Modoc, La Plata, and Poser. In addition to those named there are many smaller and less important fractures of Steward age, in part branches from the larger fissures, and in many instances sympathetic fractures of apparently limited lateral extent. The Bell fault of the Mountain Con and Diamond mines is a prominent northeast fissure paralleling the Steward faults, but of doubtful age. It is provisionally placed in the Steward system, but later developments may prove it to belong to a later period, possibly as late as the Middle faulting. The Bell fault dips 65° to the south in the upper 500 ft., 85° to the north for the next 700 ft., and 80° southerly again in the deep levels. With the general dip to the south, the north or foot-wall has moved downward relatively to the hanging-wall. (See Fig. 3.) The Bell fault is not known to contain ore other than drag from older veins. No intersections between the Bell fault and undoubted Steward fissures have been developed by mine workings. It is therefore impossible to learn their true relationship, since both exhibit the characteristic fault structure.

The Middle fault of the Mountain View mine was formerly believed to be of the same age as the Steward and it was long considered a member of that system. More recent developments prove beyond question that the Middle fault is later. It is believed that the Middle fault is, in part, a strike fault, along an earlier fissure belonging to the Steward system. This feature will be more fully discussed in connection with the Middle fault.
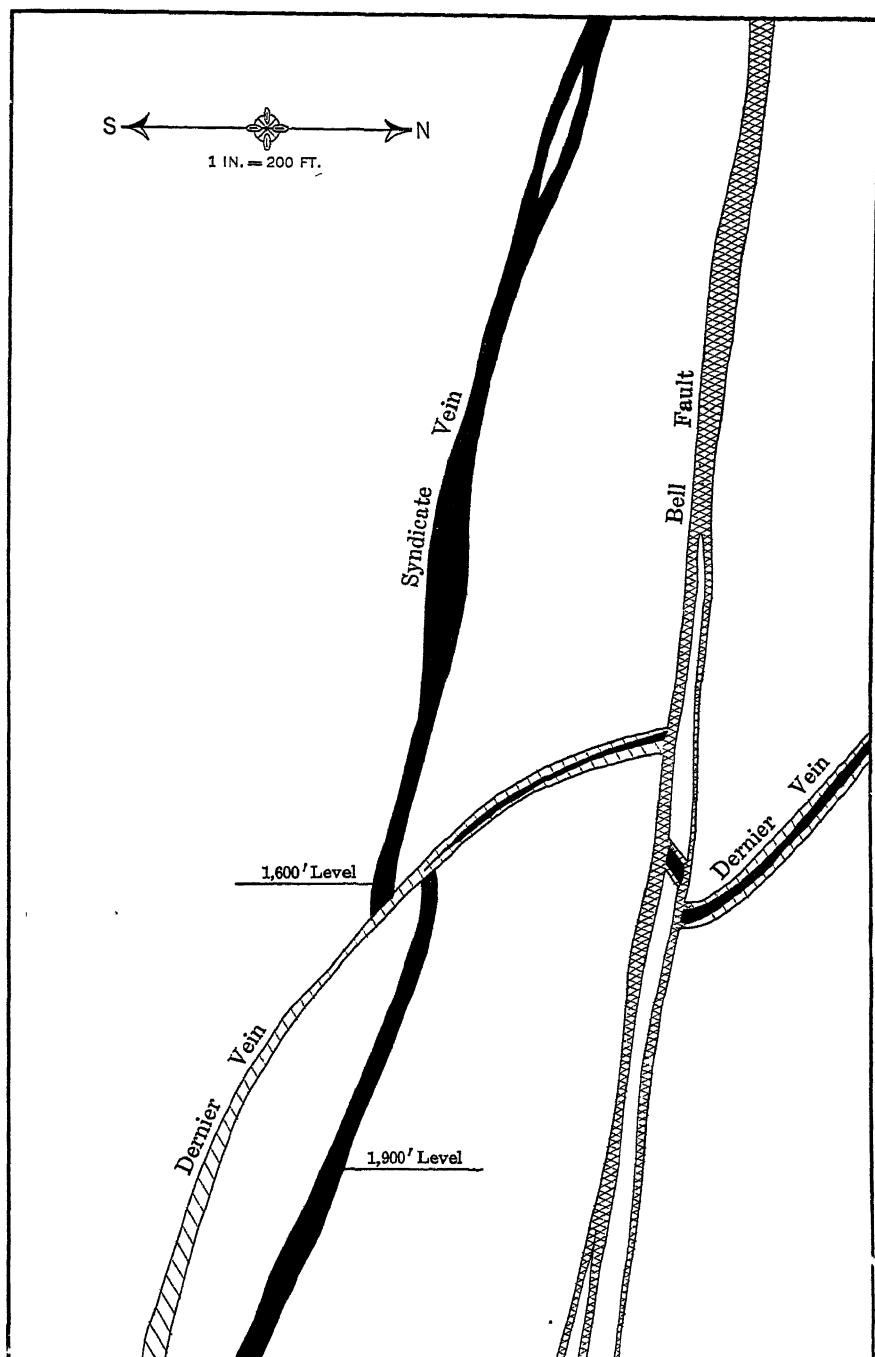
FIG. 3.—VERTICAL SECTION SHOWING DOWNWARD MOVEMENT OF FOOT-WALL
COUNTRY ALONG BELL FAULT.

The Steward faults are characteristically fissures of movement. They are narrow zones, or belts, of crushed granite accompanied by well-marked seams of attrition clay. The widths vary from a few inches up to 10 ft., depending largely on the magnitude of the faulting. The total displacement along the faults varies from 50 to 150 ft. in the important fissures. The Steward fissures are normal faults inasmuch as the hanging-wall has moved downward relatively to the foot-wall. The movement, however, did not take place downward on a line normal to the strike, but along a line within the plane of the fault making an angle of approximately 70° with the horizontal.

With but few exceptions the Steward faults carry no ore; but it appears that in certain instances they have exerted an important influence on the veins of the earlier Anaconda system. This feature will be taken up later in the description of the Steward fissures as ore producers.

The Rarus Fault.

Age.—The Rarus fault is a complex fissure of later age than the Anaconda and Blue systems of fissures. It is more recent also than the peculiar Mountain View breccia veins. It cuts and displaces the principal fissures of the Steward system, but the true age relationship of the Rarus fissure and the Bell and Continental faults is not known. In the St. Lawrence, Mountain View, and West Colusa mines the Rarus fault is faulted by a series of closely spaced south-dipping fissures called the Middle fault, as illustrated in Plate I. Northeasterly in the Leonard and to the southwest in the Moonlight mine the displacement along the Middle fault is slight and not readily recognized. Other northeast fissures in the Mountain View mine, parallel to the Middle fault, are also known to cut and displace the Rarus fault. It was first believed that the displacement of the Rarus was due largely to block readjustments locally in the Mountain View mine, where the whole body of the granite is intensely altered and there exists a multiplicity of fractures. Later evidence seems to indicate that these late movements have been more widespread than first realized, and it is not improbable that the Middle fault may mark but one of a more extensive series of movements taking place subsequent to the Rarus fault.

Geographic Position.—The Rarus fault has been opened up by numerous mine workings from the East Colusa mine on the northeast to the Belmont on the southwest, and on dip from the surface to the 2,800-ft. level. The dip is remarkably uniform throughout, varying but little from 45° to the northwest. The strike is variable, ranging from N. 30° E. to N. 80° E., the average being roughly N. 50° E. Although a well-defined compound fissure, broadly speaking, the Rarus fault in

detail is exceedingly complex.   In places there are two separate limit-
ing movement planes defined by heavy dark gray fault gouge from 1
to 8 in. thick, usually accompanied by from 10 to 30 ft. of finely
crushed altered country rock, and ore fragments where in the vicinity
of older veins.   The perpendicular distance between the two limiting
planes varies from 20 to 250 ft.   Where the distance is less than 50 ft.
the whole intervening rock mass is thoroughly crushed.   With greater
separation of the two planes the interfault ground becomes less and less
affected, so that in extreme cases, as exhibited in the Rarus mine, the
ground between the ultimate boundary fissures shows but little, if any,
effect of fault movement.   Generally, however, the included country is
intersected by parallel or sympathetic fractures crossing diagonally
from wall to wall.   In the Rarus mine the limiting fissures are distinct
and well marked and were early termed the Rarus " hanging-wall "
and Rarus ·' foot-wall " faults.

Displacement.—The displacement along the Rarus fault is greater in
the southwestern part of the district than toward the northeast; in
fact, in the Leonard and Colusa mines the fissure becomes so split up
that its identity cannot be established northeasterly from the Leonard
shaft.   It does not displace the Colusa vein in the upper levels and is
believed to die out rather suddenly in this region.   The movement
along the fault plane is not normal to the strike at all points.   The
hanging wall has moved downward, the line of movement within the
fault plane making an angle of 90° with the horizontal in the Belmont
and 60° or even less in the Leonard.   In the Belmont the displace-
ment is 350 ft., in the Rarus mine 240 ft., and in the Leonard less
than 120 ft.

An interesting feature in connection with the Rarus fault is the
shifting of the movement from hanging-wall to foot-wall, or *vice versa,*
through the medium of diagonal movement planes intersecting the
interfault granite.   This is well illustrated in Fig. 4, which shows
the faulting effect on certain veins of the Pennsylvania mine.   At
times the whole fault displacement is so evenly distributed between
the limiting boundaries by means of small movement planes that no
actual cut-off of the intersected vein occurs.   In certain instances the
movement has been entirely taken up by interior adjustments within
the granite blocks so that no visible movement planes are developed.
(See Fig. 4.)   The phenomenon of step faulting of veins is a result
of the frequent shifting of the movement from the wall through the
intervening country rock and veins.   The cause seems to be found in
the fact that the direction of downward movement of the hanging-wall
was not normal to the strike at every point.
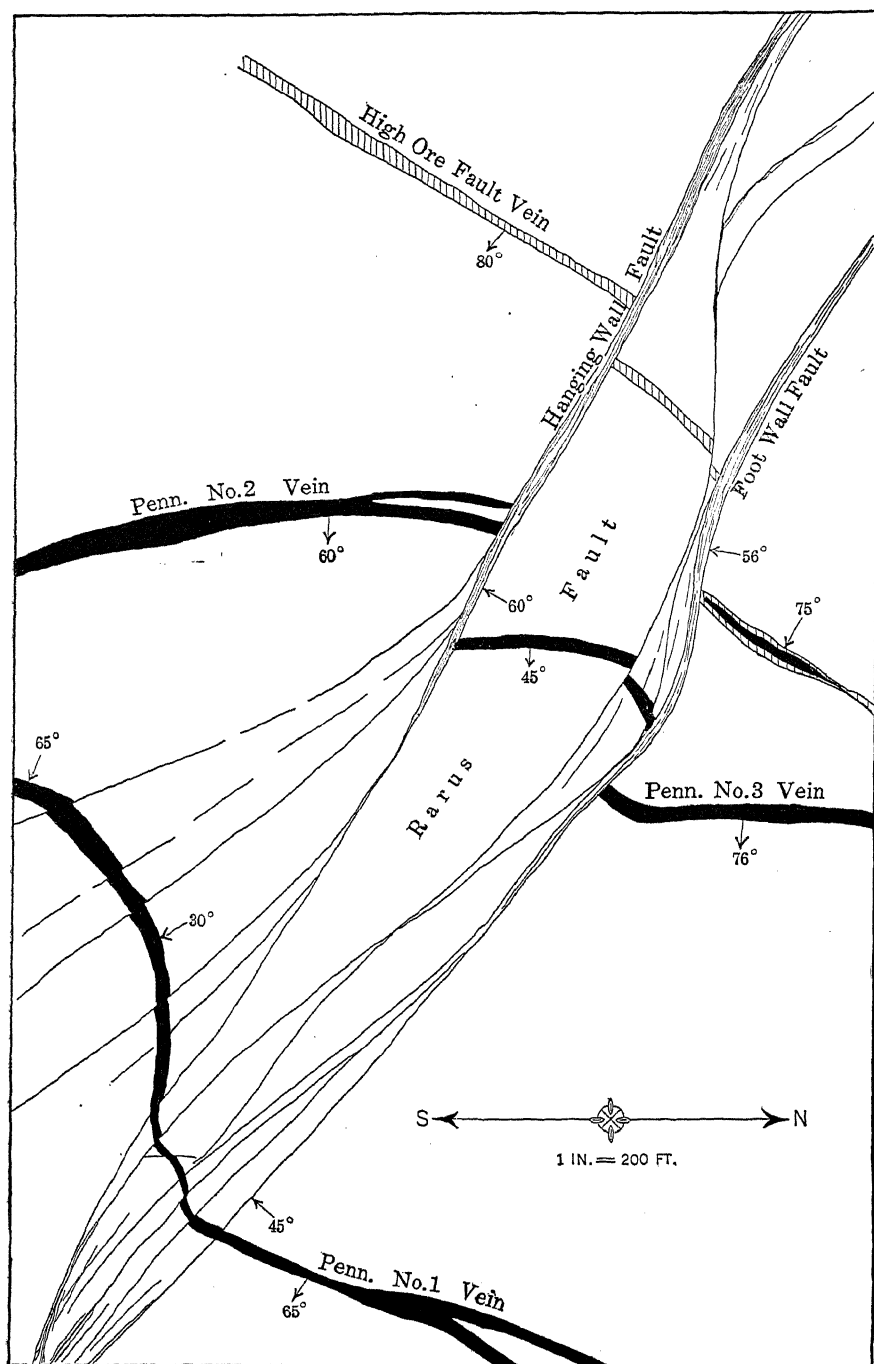
FIG. 4.—PLAN OF 500-FT. LEVEL OF PENNSYLVANIA MINE, SHOWING EFFECT
OF RARUS FAULT ON DIFFERENT VEINS.

Mineralization.—The Rarus fault has not been found to carry ore other than fragmental ore dragged from older formed veins. These included blocks or fragments varying in size from small bits, or pebbles, etc., up to large blocks or slices of veins which reach from wall to wall of the fault. In thickly veined areas it frequently happens that segments of one vein are moved within the fault to a position where the ends are brought opposite an entirely different vein lying without the fault zone. This phenomenon formerly gave rise to the belief by some geologists that no displacement occurred along the Rarus fault, for it was found to be possible in rare instances to extend a drift entirely through the fault zone on vein, although the faulted portions of three separate veins were required.

Mine developments so far indicate that the only indigenous minerals of the Rarus fault are quartz and pyrite in sparsely disseminated amounts. The influence of the Rarus fault on intersected veins will be further treated under vein descriptions.

In the Corra and Gray Rock mines a small fault fissure, known as the Corra fault, having a normal displacement of 40 ft., striking N. 75° E. and dipping 50° to the north, is believed to be of Rarus age. It cuts the Anaconda, Blue, and certain fissures of the Steward system. The age relation between the Corra and Bell faults has not been determined. The Corra fault carries no ore.

The Middle Fault.

Formerly it was believed that the Rarus fault was the latest geologically in the entire district, with the possible exception of the Continental fault at the base of East ridge. Recent mine developments, however, have disclosed the fact that the Middle fissures of the Mountain View mine represent a period of movement later than the Rarus. On account of its strike, dip, and general physical character, the Middle fault was early assigned to the Steward fissure system. It was thought that the No. 16 vein of the Rarus mine, a Steward fissure, was the faulted segment of the Middle fault, lying beneath the Rarus fault. It has been abundantly shown, however, by the recent development of many intersections of the Rarus and fissures of Middle fault age that the latter are post-Rarus.

Owing to the extreme complexity of fissuring in the region of the Mountain View mine the relation between the numerous fractures has not been satisfactorily worked out. The available evidence indicates that the Middle fault is coincident with, and, in fact, forms a strike fault along the segment of the No. 16 vein lying above the Rarus fault. (Fig. 5.)

According to the best information available, the Middle fault exhibits the greatest displacement in the Mountain View mine and vicinity, where it approximates 75 ft., with a lessening amount of movement both to the northeast and southwest. There is certain evidence indicating that the Middle fault is the manifestation of a
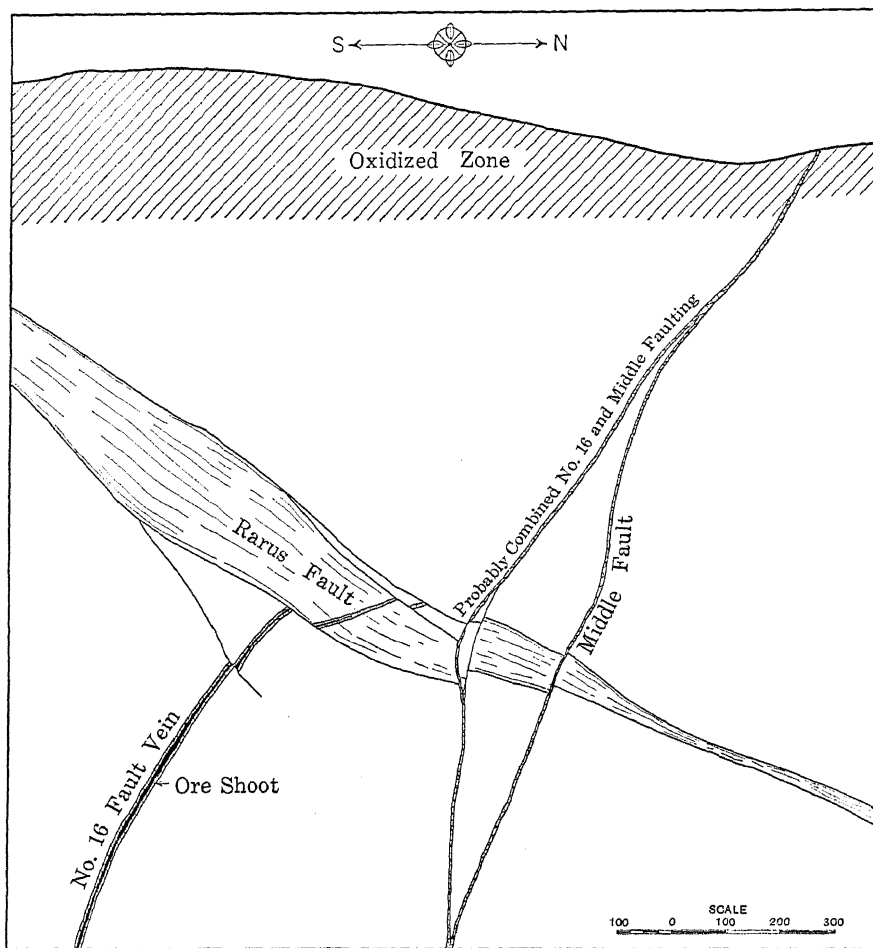
Fig. 5.—Vertical Section through the Mountain View Mine, Showing the Relation between Chalcocite Ore Shoot of No. 16 Vein, of the Steward System, and the Rarus and Middle Faults.

sagging tendency within this particular section, caused by unusual weakness in the earth's crust due to extreme alteration and intense fissuring.

The Middle fault fissure is of the same general character as the typical faults of the district. It carries no ore and exhibits no min-

eralization whatsoever. The strike is N. 65° E. with a general southerly dip, although in deep levels and to the northeast it dips slightly north. The average dip down to the 1,200 Mountain View is 65° to the south, below which level it quickly steepens to a vertical or slightly north dip, which is maintained to the deepest mine levels.

The Continental Fault.

The Continental fault lies almost entirely without the copper-producing district. It is a complex fissure zone from 200 to 1,000 ft. wide, having a general north and south strike following close to the base of East ridge. It may be seen outcropping in the Great Northern railroad cuts at Horseshoe Bend, from which point it passes southerly through the Six o'Clock, Greenleaf, Bullwhacker, Montgomery, and Amazon-Butte properties. Continuing to the south it follows closely the base of East ridge in a direction toward Nine Mile canyon.

The general dip of this fault is to the west at an angle of 75°. Within the ultimate boundaries of the main fissured zone, wide variations in dip are commonly met. In the Greenleaf mine many of the important movement planes are vertical or slightly west dipping. It appears that the western or hanging wall has moved south and downward, the amount of vertical throw being unknown, but possibly in the neighborhood of 1,500 ft. The horizontal throw is probably several hundred feet, but the evidence is not sufficient to even approximate the amount. The movement of the fault is rarely concentrated along a single plane, but is distributed over a series of roughly parallel fissures, accompanied by a variable amount of breaking or crushing of the intervening country rock. In the Six o'Clock mine a thickness of 4 ft. of tough, dry gouge, dipping 67° to the west, marks the foot-wall fissure. At this point the fault movement is much more concentrated along a single fissure than at points farther south.

No primary mineralization has been recognized in this fault. The intense alteration of the granite, so characteristic of the Blue and Steward faults, is absent. The movement planes of the Continental fault are marked by a tough gouge composed of finely comminuted, unaltered granite. The waters now found within and along the fissure are no doubt of meteoric origin and have effected but slight changes in the crushed country rock. It would seem reasonable to infer from these observed facts that the Continental fault is of comparatively recent origin, and that, at great depths, it did not reach a source of primary vein-forming waters.

The age of the Continental fault relative to the various fractures

of the district is proved by evidence obtainable in the East Butte mines, where two north and south fractures, undoubtedly as a part of the Continental complex, cut and displace veins of the Anaconda and Blue systems, and in one instance a northeast south-dipping fissure probably of Steward age. The relatively recent date of this fault is also suggested by Plate IV., a vertical section taken along the general course of the Anaconda vein showing the probable relation between the fault and the oxidized and sooty chalcocite zones.

ROCK ALTERATION.

Extensive alteration of the rocks has taken place in the Butte district. There are two principal zones or areas associated with the copper veins in which the rocks are altered to an unusual degree. These areas are closely related to the more important developments of the earliest-formed veins, or those of the Anaconda system. One of these alteration zones follows rather closely the Syndicate and other early veins in the Mountain Con, Gray Rock, and Diamond mines. In the two last-named properties the zone reaches a maximum development where intersected by many fault veins belonging to the Blue system. The outline of this zone is so irregular that it cannot be mapped with any degree of accuracy. On Plate III. an attempt has been made to indicate the general conditions. The northerly zone is seen at this elevation to be composed of three or more disconnected areas, all of which merge into one more or less continuous area at greater depths. When referring to the map, Plate III., it should be kept in mind by the reader that only the more important areas of rock alteration are shown by the cross hatching; furthermore, there has been a marked alteration of the granite along nearly every vein and fissure shown on the map, but usually not extending outward for appreciable distances from the immediate influence of the fissures, at least not extensive enough to be accurately represented on a map of this scale.

The largest and most important area of altered granite is in the vicinity of Anaconda hill. In this part of the district a great altered belt extends easterly from the Parrot mine through and including the Never Sweat, Anaconda, Mountain View, West Colusa, Berkeley, and Silver Bow mines, and within this whole area it is next to impossible to find a hand specimen of rock which has not undergone marked chemical and physical changes. As in the case of the north belt above described, there is an unmistakable close genetic relation existing between the widespread rock alteration and the Anaconda vein system, a fact more readily understood by reference to Plate III.

In these two general zones intense alteration has taken place not

only within and along the veins and faults, but the entire rock mass, whether granite, aplite, or quartz-porphyry, has been invaded by active metasomatic processes, resulting in a product differing markedly both in chemical and physical character from the original rock. The hard, dark-colored granite has been changed to a whitish or mottled gray, less firm rock, usually peppered generously with iron pyrite. The quartz-porphyry has been altered from a grayish green color to a yellowish white rock studded with glassy quartz phenocrysts. When subjected to further action by descending meteoric waters the rocks became weaker, more porous, and there is slightly more uniformity of structure. In extreme cases in newly opened workings of the upper levels the rock breaks in great slab-like pieces, the fracture planes extending directly across and without regard to joint planes.

In the areas between and outside the principal altered zones of the district the country rock generally shows but slight change, excepting within or adjacent to important veins, fissures, and shear zones, and to a less degree along joint planes and cracks. It is not uncommon to find, within general areas of unaltered rock, cracked or crushed zones in which there has been but slight chemical change in the rock. These broken zones, which may vary in width from a few inches up to several feet, appear not to have reached a source of active water circulation. Observations seem to indicate that the rock alteration is becoming more widespread as greater depths are attained. This is particularly true in the region of the Diamond, High Ore, and Leonard mines.

Causes of Alteration.

The alteration in the Butte rocks may be traced to three general causes: (1) vein formation; that is, alteration effected by solutions which were primarily responsible for the formation of the ores; (2) common hydro-metamorphism, or effects produced by the action of descending meteoric waters; and (3) oxidation.

These processes are, in fact, superimposed in part. The chemical and physical changes resulting from vein-forming processes are modified in the higher levels by meteoric waters, and a further change takes place when such altered rocks are brought by erosion or other causes into the zone of oxidation.

As already stated, the great alteration zones are distinctly associated with veins and fractures belonging to the oldest known vein period, or that embracing the Anaconda system. In regions much faulted and crushed one might naturally expect to find the greatest alteration, because of the apparently greater permeability of the broken country rock to solutions traversing the fissures. That such is not

necessarily the case is believed to be due to the following conditions:

1. The earliest ascending waters and gases were much more active chemical agents than later solutions, owing to higher temperature and pressure conditions, and to some extent, possibly, to their chemical composition.

2. The earliest fractures were fissures of but slight dislocation, therefore they were unaccompanied by impervious crushed granite and fault clay. The solutions and gases were thus accorded easy access to the wall rock at all points along the fissures. In later faults exhibiting much movement the circulating solutions were often closely confined within impervious fault-clay seams, and as a result unaltered wall rock is commonly found within a few feet of extensive ore bodies.

3. The regions traversed by the oldest fractures have been subjected to the action of vein-forming processes over a much longer period than regions adjacent to later fault veins.

There can be no question but that chemical agents have been greatly aided in their attack upon the granite and other rocks by dynamic processes. The breaking and crushing of the country rock in the Butte district has been on a profound scale. Dynamic agencies have not only furnished the avenues of travel within the rock for the gases or highly heated water, but, through accompanying crushing and mashing, the rocks are made more susceptible to attack by solutions. The true nature of the changes in the rocks, other than crushing, wrought by dynamic agencies, has been in a large measure obscured by subsequent metasomatic processes accompanying vein formation.

Vein-Forming Processes.—The alteration from this cause took place along and outward from fractures which acted as channels for the uprising solutions. These waters, of deep-seated origin, traversed not only the main trunk channels and associated fractures, but in highly fissured areas they followed also the joint planes, and penetrated, by slow stages no doubt, the whole mass of the granite in the more highly fissured areas. The remarkable activity of these thermal processes is evidenced by the development of the extensive altered zones accompanying the oldest fracture systems.

The initial stage of these changes, which were metasomatic in their nature, seems to have been the development of chlorite accompanied by the formation of pyrite. The uprising thermal waters or gases attacked first the iron silicates, augite, hornblende, and biotite, forming chlorite, epidote, secondary silica, and iron pyrite. Pyrite was developed also from the iron of the magnetite, the sulphur in this case, as in the former, being furnished by the attacking thermal waters, in

which it existed as hydrogen sulphide or as an alkaline sulphide. Plagioclase and orthoclase feldspars give way to continued attack, resulting in the formation of sericite and secondary silica. The result of the continued action of these metasomatic processes upon the granite has been the development of "pyritized" granite. The early-formed chlorite and epidote largely disappear; practically all of the iron of the original granite is converted into pyrite; and the feldspars are broken up into sericite and quartz. "Pyritized" granite, therefore, where representing the extreme development of the sericitic stage, consists principally of sericite, quartz, and disseminated pyrite.

Since the abundant fractures of the various systems form the medium through which thermal waters, or vein-forming solutions, traverse the body of the rock, the alteration necessarily proceeds, generally speaking, outward from these channels. The alteration is therefore usually more intense immediately along and within the fractures or fracture zones. Vein-forming processes have altered large areas of rock within and tributary to the Anaconda fissures; the Blue veins show less alteration of the wall rock than the Anaconda fissures, and similarly the Steward faults exhibit less alteration of the wall rock than the Blue veins, although in every case alteration has been intense within the fissures themselves.

Charles T. Kirk's[5] studies indicate that these various alteration phases are seldom free from the presence of kaolinite, a mineral of uncertain origin. It may be here stated that the relative quantity of kaolinite present in the deeper levels is insignificant when compared to the amount present near the surface, where it is known to result from the action of cold meteoric waters on pyritized or sericitized granite. There seem to be excellent reasons, to be later given, for believing that a large part of the present-day existing ground-water, even to great depths, is of meteoric origin, carrying appreciable quantities of iron sulphates and sulphuric acid. In the presence of such waters a slight development of kaolinite might be reasonably expected, while an important actual movement of these waters need not be inferred.

Common Hydro-Metamorphism.—Cold meteoric waters penetrating and passing downward through areas of unaltered Butte granite have effected but slight changes. The chemical activity of such solutions has apparently been slightly increased where preceded by crushing, and notably increased where preceded by rock alterations caused by vein-forming waters. The action of meteoric waters has, therefore,

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[5] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, p. 60 (Jan., 1912).

been most intense in veins and in the great alteration zones associated with the Anaconda fractures earlier described. The reasons for this increased activity in veins and in regions of altered granite are many. The oxidation of the pyrite of the veins and the disseminated pyrite of the altered granite results in the formation of iron sulphates and free sulphuric acid, which readily attack the already altered granite below, converting the sericite largely to kaolin. This process develops greater porosity in the granite, thus affording a more ready passage for the surface waters to greater depths.

The most noticeable effects upon the pyritized altered granite of descending meteoric waters have been kaolinization and chalcocitization, accompanied by greater porosity. In general, the ordinary meteoric waters have had no noticeable chemical effects upon the normal Butte granite at depths greater than the vertical thickness of the oxidized zone, which is seldom more than a few feet in unaltered granite.

Depth Reached by Meteoric Waters.—The maximum depths reached by meteoric waters in the Butte district cannot be definitely determined. It is probable that they have descended to greater depths than any yet reached by mine workings. A study of the physical and chemical changes that have taken place in the veins and country rock known to be due to the presence of waters of meteoric origin, offers the only criteria for a partial solution of this problem.

The oxidized zone, resulting from the oxidizing influence of meteoric waters, varies in depth from 10 to 500 ft., an exceptional case in the Mountain View mine measuring over 900 ft. from the surface. The average depth, however, along the Anaconda vein is not more than 300 ft. That surface waters move downward to depths much greater than the lower limit of the oxidized zone is proved by the occurrence of an abundance of minerals in the veins and country rock known to be of secondary origin. Of these, chalcocite and kaolinite furnish the most reliable indicators of meteoric water influence. In veins and pyritized granite wall rock below the zone oxidation, the processes of chalcocitization and kaolinization go hand in hand, with the important difference, however, that while chalcocitization is always accompanied by kaolinization, the reverse is not necessarily true. Kaolinite, probably resulting from meteoric water action, is found at much greater depths than undoubted secondary chalcocite, and from this fact it is believed that down-seeping sulphate solutions continue to act on altered granite, forming kaolin, long after the last trace of copper has precipitated out as chalcocite in the zone of secondary chalcocite at higher levels.

Since primary chalcocite occurs in great abundance in the Butte veins, the mere presence of mineral chalcocite is not indicative of the presence of meteoric waters. The presence of the "sooty" chalcocite, however, which is known to be a product of descending meteoric waters, may be generally taken as proof of surface water action.

As a reliable means of determining more or less accurately the amount of vertical descent of meteoric waters, secondary chalcocite loses much of its importance when it is considered that the depth of the zone of secondary chalcocite is dependent upon many variable factors such as topography, mineralogical and physical character of the vein, etc. The depth of the zone of secondary chalcocite varies from 50 to 200 ft. in the fault veins to a maximum of 1,200 ft. in the veins of the Anaconda system. With the aid of a reliable method for distinguishing between primary and secondary chalcocite when in the massive form, the value of this mineral as a criterion of descending sulphide enrichment will be greatly increased.

If kaolinite is characteristically a product of meteoric water action, it is safe to conclude that surface waters have descended to depths greater than any yet reached by mine shafts. However, if, as Gregory[6] holds, kaolinite may also result from hydrothermal action at great depths, the small quantities of kaolinite present in the deep Butte levels may not be properly regarded as proof of the presence of waters of meteoric origin.

Oxidation.—Oxidizing processes acting upon veins and altered granite tend to transform the sulphides into oxides and native metals. The results of these processes acting on Butte ores and rocks are briefly described in the chapter following.

Superficial Alteration of the Butte Veins.
Outcrops of Copper Veins.

The mantle of "wash," or débris from disintegration and weathering, often masks the intersection of the veins with the surface of the bed rock. This condition is especially prevalent in areas of intense granite alteration, notably eastward from the Parrot and Moonlight and in the vicinity of the Diamond mine. The thickness of the surface wash varies from 2 to 10 ft., although an exceptional thickness of from 200 to 400 ft. occurs in the valley in the vicinity of the Pittsmont smelter. (See Plate IV.) This unusual thickness, however, has probably resulted from certain conditions brought about by the Continental fault movement, rather than from natural processes of erosion and decay.

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[6] Gregory, J. W., Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 7, p. 680 (Oct.–Nov., 1910).

A number of copper veins, notably the Anaconda, Syndicate, and Colusa, have more prominent outcrops characterized by unusual amounts of strong iron-stained quartz and vein matter projecting well above the wash. As a general rule, however, the position of the copper vein outcrops, especially within the alteration zones, cannot be accurately determined without the aid of shafts or tests pits, although the general location and direction may be known by the presence of detached fragments of the veins, or "float." The degree of prominence of vein outcrops appears to be governed largely by their mineralogical content and physical character, and also by the nature of the inclosing wall rock.

The outcrop of a typical copper vein of the Anaconda system is marked by altered granite, quartz, and oxides of iron. There may also be present a small band of oxidized clay or crushed granite within or along the vein. The granite included within the vein boundaries or adjacent to the vein is irregularly seamed and stained with iron oxides. The quartz commonly exhibits the well-known honeycomb structure.

The outcrop of a fault vein of the Blue or Steward systems consists of a slightly iron-stained mass of soft, crushed and altered granite with one or more seams of fault clay of a blue-gray or yellowish color. Where a fault-vein ore shoot outcrops, the composition is very similar to that of the Anaconda veins, with the possible addition of one or more well-defined bands of fault clay.

Almost without exception the copper veins are practically barren of copper at the outcrop and in the zone of oxidation below. There are no visible copper minerals, and it rarely happens that an assay of the oxidized material yields more than a trace of copper. (Compare Table I., following.) Some exceptions, however, may be noted. Small quantities of carbonates and oxides of copper were found in the outcrop of the Gagnon-Parrot vein, also in the Syndicate vein and others. The discovery shaft of the Mountain Chief claim, located on the southeasterly extension of the Jessie vein, was sunk in a rich body of red oxide of copper, running high in silver. The oxidized zone, however, proved to be very shallow in vertical extent, the rich oxide ore changing abruptly to chalcopyrite at less than 30 ft. in depth. (See Plate V.)

Taking a broad view of the entire copper producing area, it may be said with emphasis that there is but little, if any, evidence of a positive character to be found in the outcrops or the oxidized zones of the Butte veins to indicate the existence of copper in commercial quantities at greater depths.
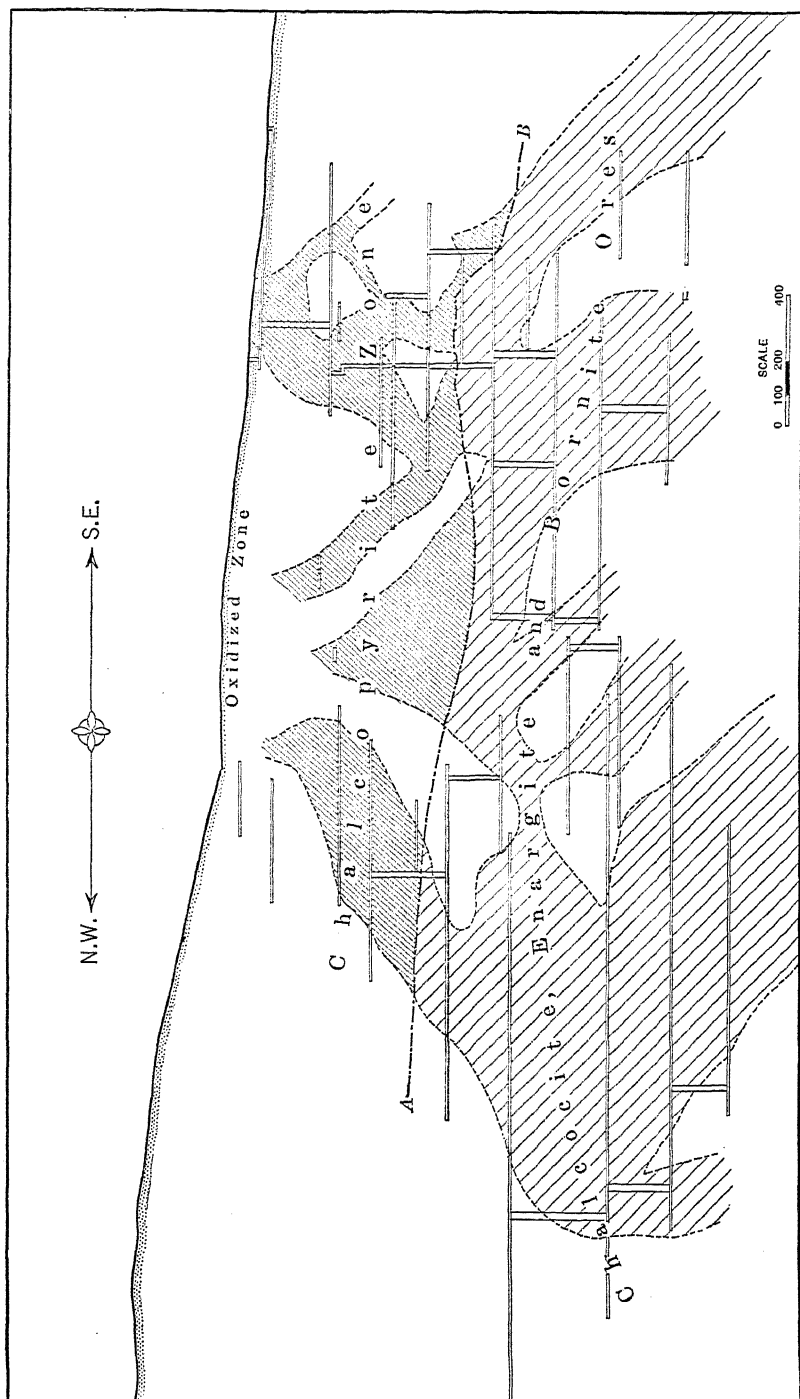
PLATE V.—LONGITUDINAL PROJECTION OF A PORTION OF THE JESSIE VEIN, SHOWING ARRANGEMENT OF ORE SHOOTS.
Line A–B marks approximately the dividing line between chalcopyrite and rich chalcocite-enargite ores below. Both above and below line
A–B quartz and pyrite are abundant as gangue minerals. Above line A–B rhodochrosite is commonly present.

The following table gives the analyses of a series of samples taken from oxidized portions of producing copper veins:

TABLE I.—*Analyses.*

	I.	II.	III	IV.	V.	VI.	VII.	VIII.	IX.	X.	XI.	XII.	XIII.
SiO₂	48.0	84 2	65 5	41 1	79.2	67.4	71 7	68.3	73.3	26.6	58.4	51.9	56.2
Fe	4 6	3.3	15.0	30.6	5.9	12.3	8 9	10.2	5.4	19.8	12.5	16 5	17.6
Al₂O₃	25.3	5.9	4.3	5 4	5.5	7.4	4.1	6.4	11.5	25.9	5.5	8.5	4.7
Cu	0.05	tr	0.12	0.13	tr	0.15	0 05	0.05	tr	0 05	tr	0 05	0.20
As₂O₃	1.71	0.14	0.79	0 52	0.22	0.38	0.32	0.56	0.38	0 88	5.24	1.29	2.29
Zn	tr.	tr.	tr.	tr.	tr.	tr.	tr.	tr.	tr.	tr.	tr	tr.	tr.
Ag, oz per ton	17.7	3.7	5 8	1 0	tr.	4 1	3 1	10 9	1.2	1.1	2.8	4.2	12.0
Au, oz. per ton	0 05	0.008	0.005	0.005	0 07	0.01	0.015	0 02	0 01	0 007	0.03	0 01	0.04

I. Lloyd tunnel 250 ft. below outcrop of vein.

II. Well-defined quartz vein 5 ft. wide; sample taken 10 ft. below outcrop.

III. Oxidized quartz vein of Anaconda system; sample taken 15 ft. below outcrop.

IV. Soft dark red oxides along fault clay; sample taken 15 ft. below outcrop.

V. Sample of 4-ft. siliceous vein taken 20 ft. below outcrop.

VI. Oxidized enargite-chalcocite ore taken 2 ft. above sulphides and 400 ft. below outcrop.

VII. Oxidized vein; sample taken 20 ft. above sulphides, St. Lawrence mine, and 100 ft. below outcrop.

VIII. Anaconda vein, Mountain View mine; much enargite in ore below; sample taken 500 ft. below outcrop.

IX. Anaconda system, South vein; taken above secondary chalcocite ore, 260 ft. from outcrop.

X. Soft oxidized clayey vein, 20 ft. below outcrop.

XI. Blue vein system; taken above enargite ore, 360 ft. below outcrop.

XII. From fault vein enargite ore, Mountain View mine, 450 ft. below outcrop.

XIII. From south vein, Mountain View mine, 600 ft. below outcrop.

The above samples are from veins lying well within the alteration zone in the vicinity of the Rarus mine. The most significant feature is the high content of As₂O₃, indicating that not all of the arsenic is extracted from the vein in the process of oxidation. Further investigations are now under way to determine if possible the relation between the arsenic present in the oxidized to the total arsenic in the unoxidized vein below.

Outcrops of Manganese-Silver Veins.

In marked contrast to the ill-defined copper-vein outcrops are the bold projecting outcrops of the manganese-silver veins, which may be traced for hundreds or even thousands of feet over the surface. Many of them, like the Rainbow, Silver Lick, Ancient, Emma, and numberless others, project in bold relief from 1 to 10 ft. above the ground surface. Where they do not project above the wash, their presence is generally indicated by an abundance of characteristic float rock.

The outcropping portion or the oxidized zone of a typical manganese-silver vein consists chiefly of quartz and oxides of manganese and iron. As an original mineral of the unoxidized vein, pyrite is not as abundant as in the copper veins, nor is it so universally present.

The Black Rock vein, an important producer of zinc, is a member of the manganese-silver vein series belonging, it is thought, to the Anaconda fracture system. The developments in the Black Rock vein show that many of the great sphalerite ore bodies first appear several hundred feet below the surface, and that the upper portion of the vein is composed principally of quartz and rhodochrosite. The outcrop is mainly quartz and oxide of manganese.

In the outcrops and oxidized zone of the zinc-producing veins the zinc is entirely removed in the processes of oxidation, and not more than a trace remains to indicate the presence of zinc at greater depths.

Oxidation and Disintegration of the Granite.

In the great zones of altered granite associated with the copper veins there are but few, if any, actual outcroppings of solid granite or bed rock, the latter being effectually concealed by the covering of disintegrated rock. This surface wash, or débris, consists of iron-stained altered granite in the form of angular fragments, and finer incoherent grains, together with fragmentary oxidized vein quartz, aplite, and quartz-porphyry. The clusters of rounded boulders so characteristic of the weathering of normal granite are entirely absent.

The unaltered granite is apparently more resistant to the action of atmospheric agencies than the altered granite, and owing to this fact veins or fault outcrops found within normal granite areas are conspicuous as belts or zones of finely disintegrated rock, readily traceable over the surface. Generally in the manganese-silver vein area and all of that portion of the Butte district lying to the north of the Speculator, Tuolumne, and Corra mines, there is but little alteration of the granite, excepting within and along the vein or fault fissures. Under these conditions the positions of the veins and faults are indicated by smooth surfaces, while the intervening areas of normal granite are marked by

the presence of rounded granite boulders, a feature characteristic of normal granite weathering over the whole Boulder granite area.

Zone of Oxidation.

By the expression " zone of oxidation " is meant those portions of the veins and country rock which have been oxidized through the action of descending ground-waters. It embraces in vertical extent all of the veins and country rock lying above the irregular boundary plane marking the upper limit of the sulphides. (See Plate IV.) The depth of the zone of oxidation in the Butte district is extremely variable in different localities, and in exceptional instances wide variations occur within small areas. For example, in the Mountain View mine, No. 4 vein, the first sulphide ore was met at 250 ft. below the surface, or 150 ft. above the first level, while at a point 400 ft. south of the main shaft the South vein is oxidized for a short distance along its strike to a depth of over 900 ft. below the surface. Extreme local variations of this nature, however, are not the rule. The depth of the zone of oxidation in a particular locality is largely dependent upon the character of the veins and the inclosing country rock (see Plate II), to a greater extent, in fact, than upon the topographic features. The deepest oxidation is found in the great zones of altered granite described on pages 30–31, inclusive, where it runs from 100 to 400 ft., averaging in the neighborhood of 250 ft. (See Plate IV.) Under similar conditions of granite alteration the heavily mineralized veins of the Anaconda system show deeper oxidation than the fault veins, owing to the greater permeability of the quartz vein to downward-seeping waters and to the greater abundance of pyrite, the source of the all-important oxidizers, ferric sulphate and sulphuric acid. In regions unaffected by the widespread hydro-thermal processes accompanying the early vein formation, the depth of oxidation along the veins of the Anaconda system averages about 75 ft., and that of the fault veins 20 ft., with frequent variations in both classes of veins.

Examined from the surface downward the oxidized portion of a copper vein will show but little change in physical character and mineral composition between the outcrop and the sulphide ore below. The line of separation marking the change from oxidized to sulphide ore is extremely sharp. Above this contact plane there is no mixture of oxides and sulphides, the oxidation is complete. The entire change, as shown at any single cross-section of a vein, takes place within a vertical distance of a few feet. Frequently, near the upper limits of the sulphide ore, the proximity of the zone of oxidation is indicated by slight changes in the relative abundance of certain secondary minerals,

but in the case of the oxidized vein there is seldom any change indicating nearby sulphides.

In the quartz-pyrite veins of the Anaconda system within the copper-producing area, the appearance of sulphides at the lower limit of the zone of oxidation almost always means the beginning of commercial copper ore. This feature of the heavily mineralized veins is in striking contrast to the fault veins of the Blue and Steward systems, where the ore shoots are often separated vertically by hundreds of feet of barren unoxidized vein matter which begins at the base of the shallow oxidized zone below outcrop. (See Figs. 6 and 6A.)

Unusual conditions of oxidation are found in the Bullwhacker, Butte & Duluth, and adjoining properties, situated along the line of the Continental fault outcrop east of the Pittsmont mine. It appears that the unoxidized granite of that section carries a small percentage of disseminated chalcopyrite. The shearing and crushing along the fault zone caused a rapid disintegration of the granite under the action of atmospheric agencies, during which process the copper of the chalcopyrite was oxidized and carried downward by surface waters and redeposited as chrysocolla, red oxide, or as a carbonate. The granite exhibits but slight alteration. The resulting ore is therefore a green-stained granite with the more important accumulations of copper carbonate along the cracks and joint planes. As mined the ore runs from 1.5 to 4 per cent. The absence of pyrite in the original disintegrating granite explains the formation of ore of this character. In the process of oxidation of chalcopyrite, insufficient ferric sulphate or sulphuric acid is formed to prevent the formation of the insoluble oxides, silicates, and carbonates of copper. Under such conditions copper migrates but short distances.

GROUND-WATER.

Extensive mine developments have disclosed many interesting facts concerning the distribution of the underground waters in the veins and rocks of the Butte district. In areas of intense rock alterations there is approximate saturation of the rocks and veins, a feature in striking contrast to areas of normal granite, where alternate wet and dry zones are the rule. In the great zones of rock alteration not only are the veins wet, but the intervening country carries water as well. The granite in one part of a cross-cut through an area of altered granite may be wetter than at other points, but no part of it will be absolutely dry. The inclosed veins and faults may or may not be wetter than the granite, the more important trunk channels not being readily recognized as in the zones of unaltered granite.
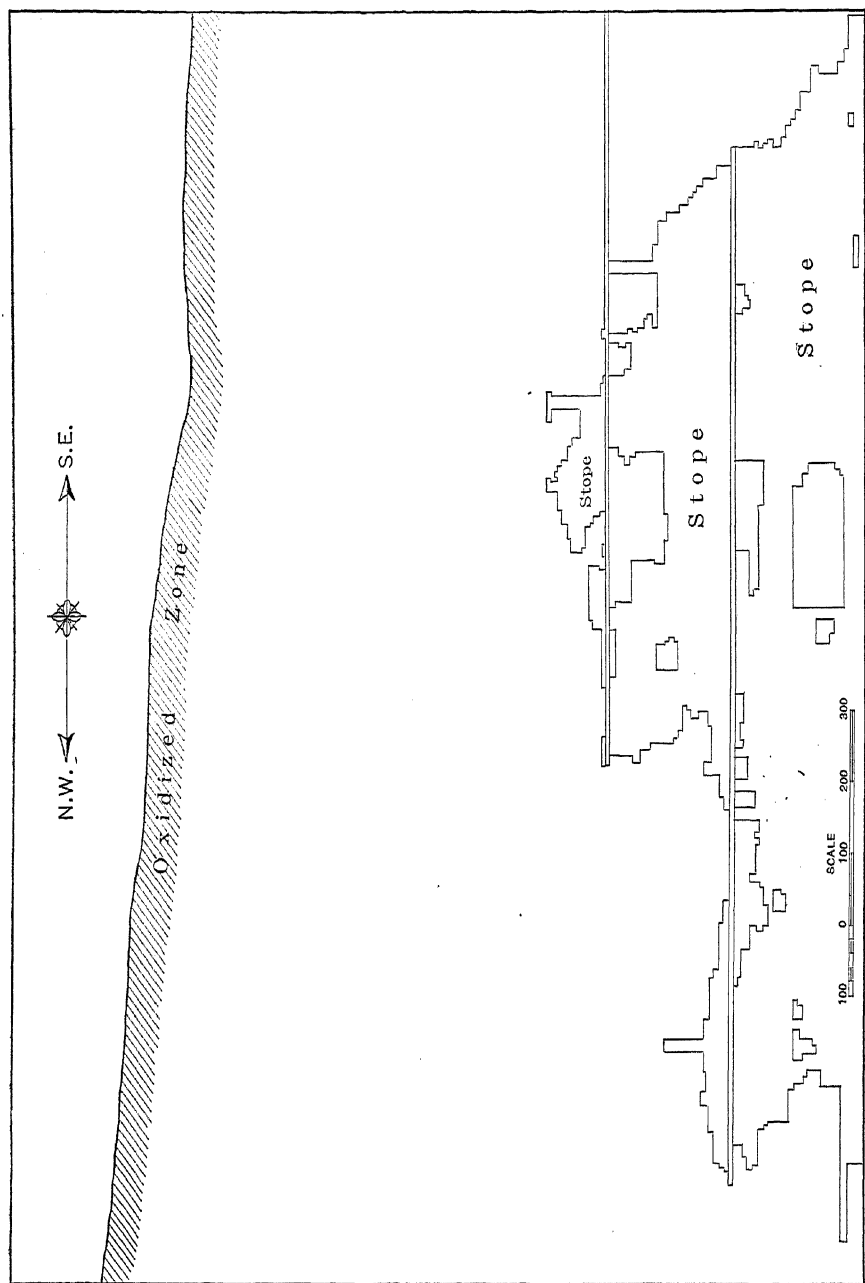
FIG. 6.—LONGITUDINAL SECTION OF THE EDITH MAY VEIN OF THE BLUE VEIN SYSTEM, SHOWING THE RELATION BETWEEN THE SURFACE, OXIDIZED ZONE, AND THE BEGINNING OF COMMERCIAL ORE BELOW.
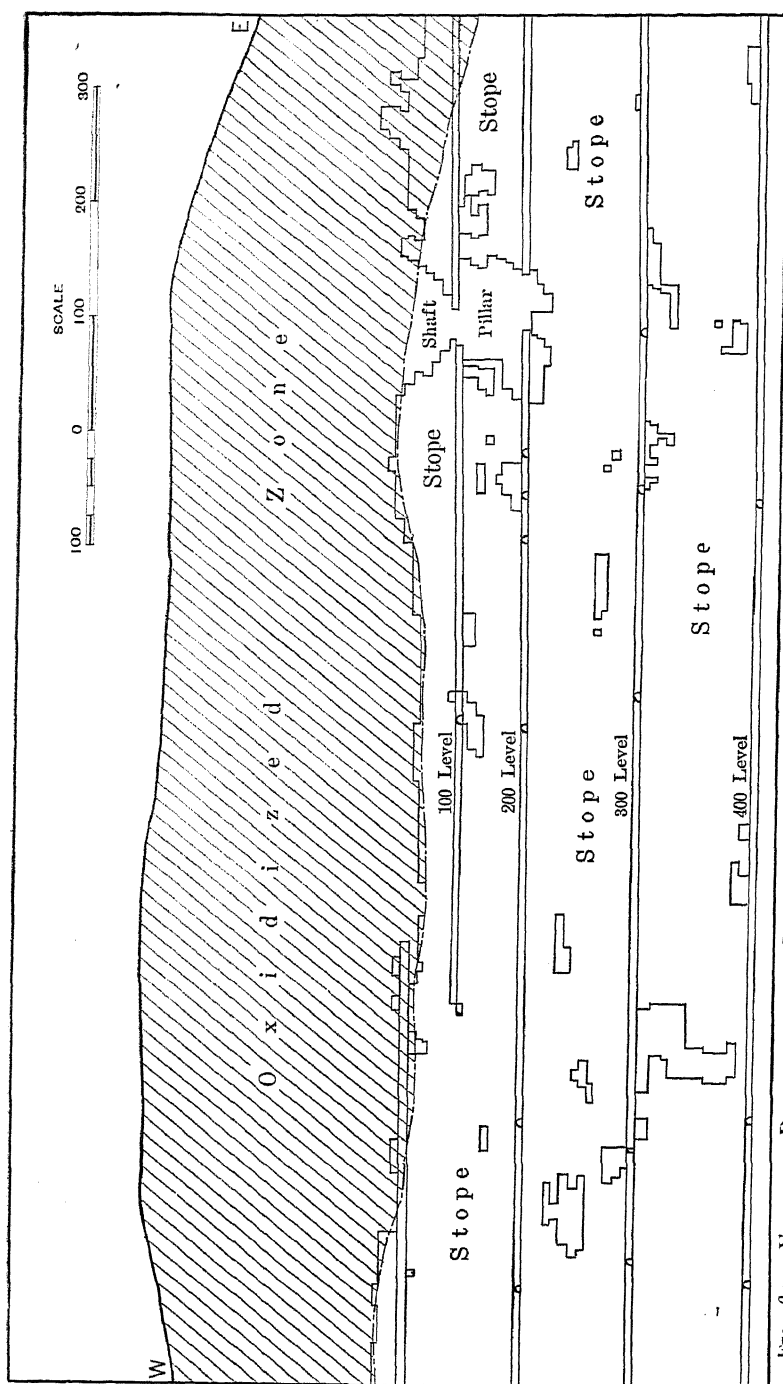
FIG. 6A.—VERTICAL PROJECTION OF A PORTION OF PENNSYLVANIA VEIN, A MEMBER OF THE ANACONDA SYSTEM, SHOWING THE RELATION BETWEEN THE OXIDIZED ZONE AND THE COMMERCIAL ORE BELOW.

In this connection it is not implied that in areas of altered granite water moves freely through the actual rock. It is believed, however, that the water does follow rather freely, comparatively speaking, the numerous joint planes, from which it spreads outward into the altered solid rock much more readily than in the case of unaltered granite. The saturated condition of the altered granite is therefore due to the ease with which the water finds its way along every insignificant joint plane or crack. The altered areas are more fractured than the areas of normal granite, because they were, in the first place, zones of the most intense early fissuring. This early fissuring being followed by marked alteration, a weakening of the rock naturally resulted. Because of this weakening, the faults of later systems commonly spread out over wider zones and were accompanied by greater crushing than in areas of normal granite.

In areas of normal or unaltered granite the ground-water is confined to the definite channels or water zones afforded by shear zones, cracks, faults, or veins. Generally the unaltered granite between the main circulation channels is dry. From these facts it appears that the unaltered granite is quite impervious to ordinary underground waters under normal conditions of temperature and pressure.

In the early days of Butte mining the water level was encountered in the shafts in the region of the contact between the oxidized and sulphide ores. This was true, however, only in areas of altered granite or where the opening was made within a vein. A shaft sunk in the normal unaltered granite between faults and veins, perchance missing a water course, might extend to great depths as a dry shaft. On the other hand, a shaft put down in the altered granite areas became a wet shaft as soon as the general water level was reached, whether a vein was encountered or not, the whole mass of the altered rock being practically saturated.

It has been repeatedly observed that the well-developed fault fissures do not serve as the most important channels for the movement of the present day ground-waters, although the influence of the fault movements in determining the distribution of these waters has been very marked. For example, the Bell, Rarus, and Middle faults seldom contain water in unusual quantities; on the contrary, they are often dry. They have broken and cracked older veins, making them quite permeable to solutions. Of much greater importance as water carriers are the veins of the Anaconda system, fissures, cracks, and shear zones of relatively slight displacement. The Anaconda veins are frequently of a porous nature and contain many cavities, vugs,

and water courses. They are especially good water carriers where cracked or broken by late fault movements.

The ore-bearing faults of the Blue and Steward systems are alternately wet and dry along the strike. It is believed that originally the uprising solutions spread out along these fissures throughout the entire length, but continued earth movements along the fissure planes developed zones of impervious clay and crushed granite, and the circulating solutions were directed along the more open zones now occupied by the ore shoots. The dry zones have remained so, and the waters now found in association with the fault veins are encountered in the more porous ore shoots, or in cracks and broken granite of the walls of the fissure. Frequently the ore shoots contain no water, and it is probable that they have been dry ever since the close of the period of thermal water activity, when the ores were deposited.

The Mountain View breccia veins are practically impervious to water. It appears that they were never connected with active uprising waters, although in part, at least, they are older than the ores of the Steward fault veins. The breccia filling of these cracks is often horizontally bedded, showing characteristic water action. It is probable, therefore, that during the period of filling, the fissures were filled with water practically stagnant.

Source of the Ground-Water.

There are two possible sources of the water now found in the rocks and veins of the Butte district: (a) uprising waters of deep-seated origin, and (b) meteoric water derived principally from atmospheric precipitation in the form of rain or snow.

Primary or Juvenile Waters.—The primary ores of the district were deposited from ascending waters presumably of deep-seated origin. The deposition of primary minerals and ores apparently stopped some time prior to the Rarus faulting, but it is not known how much longer ascending waters continued to traverse the channels of circulation after ore deposition ceased. It is reasonable to assume that during the period of primary ore formation the ascending waters were very active and plentiful and they constituted the bulk of the waters then occupying the veins and adjacent granite. A rapid waning in the activity of the ascending waters took place, no doubt, at the close of the period of primary ore deposition, after which time downward-seeping water of meteoric origin more and more predominated, until at the present day it is extremely doubtful whether any appreciable quantities of the mine waters found above the 3,000-ft. level have been derived directly from deep-seated sources.

Meteoric Waters.—The annual precipitation in the district varies from 12 to 20 in. Of this amount a certain part evaporates, some finds its way to the streams as run-off, and the remainder sinks into the earth and by means of cracks, veins, porous rocks, etc., reaches to considerable depths, forming the ordinary ground-water. On account of the extreme porosity of the surface wash covering the district, an unusually large proportion of the rainfall sinks to the contact between the wash and the solid rock below. This factor is offset in a large measure by the steepness of the slopes, permitting a proportionately large run-off.

Underground Circulation.—Actual underground observations of the action of water in the veins and rocks as disclosed by mine openings are not especially important in determining the immediate source of the ground-water. Where an opening such as a cross-cut or drift penetrates undrained water-bearing rocks or veins a pervasive dripping from the top or roof of the opening, due to gravity, is certain to follow, regardless of the original source of supply. Irregular cracks and water courses, of which there are many, both in veins and in the hard country rock, may appear in the top, side, or bottom of the opening, exhibiting either a downward or an upward flow of water according to whether the water channel first appears in the bottom or in the back of the opening.

The flow of water encountered in a new bottom level decreases rapidly, but does not entirely cease until additional openings are made at greater depths in the immediate vicinity. While this observed condition indicates a water-soaked condition of the rock and included veins or fissures, it does not afford a reliable clue concerning the immediate source of the water. Whether the source of supply is from above or below, the flow from the opening will be maintained until a certain rock volume surrounding the opening is drained. The rate of flow, however, will vary from a maximum when the opening is first made, to a certain minimum after the opening is made, the length of time elapsing depending on many factors, such as volume of ground to be drained, porosity of the rock, etc.

The fact that the water flow decreases rapidly from newly opened ground indicates that the normal ground-water of the district is extremely sluggish, if not practically stagnant, particularly in the deep levels. In higher levels incontrovertible proof of a certain amount of ground-water activity is afforded by the zone of oxidation and, in slightly deeper zones, by the presence of sooty chalcocite, an undoubted product of descending water action. After the altered rocks and veins of the great altered zones and the veins of the unaltered gran-

ite areas become waterladen, it is certain that a small supply only need be added to maintain saturation. The fact must be kept clearly in mind that the water flow encountered by mine openings penetrating new and undrained country in the Butte district does not represent the rate at which water is being supplied, but instead merely the rush of water into the opening caused principally by force of gravity.

Ground-Water Largely of Meteoric Origin.—The facts and conditions heretofore outlined seem to indicate a meteoric origin for the greater part of the present mine waters of the district. The frequent occurrence of dry ore shoots in deep levels, known to have been saturated at a former time with ascending water, indicates that the period of intense activity of uprising water has long ago ceased. In certain instances, such as in the Tramway deep levels, where unusually high water temperatures are noted, one is led to suspect a possible dying ember of a former active hot-water circulation. There are no active hot springs and no waters have been encountered in the mines to which one might ascribe with any degree of certainty a deep-seated origin. The series of temperature observations given below, however, indicate that the waters encountered in areas of intense granite alteration are slightly hotter than waters in unaltered areas at corresponding elevations. The waters of the Tramway mine traverse intensely altered granite zones, and they are, furthermore, in close proximity to certain of the high-grade primary chalcocite ore shoots of the No. 16 vein belonging to the Steward system, known to be the most recent of the primary copper ore bodies.

TABLE II.—*Temperature Observations on Butte Mine Waters.*

Mine.	Mine Level.	Elevation above Sea Level.	Temperature Degrees F.	Remarks.
Tramway..	2,800	3,400	102	Water encountered in shaft station bottom level.
Tramway.,...........	2,200	3,571	100	Water in drift newly opened.
Tramway.............	2,000	3,770	104	Drift on north-south vein near No. 16 fault vein.
Tramway.............	2,000	3,770	100	Water from cross-cut.
Tramway.............	1,700	3,917	94½	Drift on vein, new work.
St. Lawrence.......	2,800	3,395	97	Average of four observations at different points bottom level.
Pennsylvania.......	1,800	3,940	92	Water from diamond-drill hole in altered granite area.
Pennsylvania.......	1,800	3,940	90	Water from diamond-drill hole partly in altered granite area.
Pennsylvania	1,800	3,940	87	Water from diamond drill hole outside of altered granite area.
Pennsylvania.......	1,800	3,940	94	Water in face of drift on No. 1 vein in altered granite area.
Gagnon	1,900	3,955	88	Average of 8 observations in new cross-cut in normal granite area.
Original...............	2,800	3,400	99	Water in face of drift on Gagnon-Original vein.
Original...............	2,800	3,393	93	Water in face of drift on Gagnon-Original vein.
Diamond.............	2,800	3,393	95	General average of observations, bottom level.
Diamond.............	2,600	3,590	94	General average of observations.
Badger..............	1,800	4,430	76	New drift in new territory on east-west vein.
Badger..............	1,800	4,430	74	Water running from diamond-drill hole in area of normal granite.

The Tramway, St. Lawrence, and Pennsylvania mines are within the great altered zones of Anaconda hill. The Gagnon and Original are in an area of normal granite, although a prominent altered belt is associated with the main Gagnon-Original vein. (See Plate III.) The Diamond area shows a less general altered condition than the main altered zone. In the Badger mine granite alteration is prominent only within and along the veins.

MINERALOGY OF THE VEINS.

It is not intended in this chapter to describe in detail all of the minerals found in the district, but to note only those of importance in connection with the ore deposits.

The important copper minerals of the Butte ores, named in order of their relative abundance, are; chalcocite, enargite, bornite, chalcopyrite, tetrahedrite, tennantite, and covellite. Of the oxidized products, chrysocolla, malachite, cuprite, and native copper are the most common, but taken as a whole they have contributed but little to the total copper produced. The gangue minerals are principally quartz and pyrite, occurring in about equal amounts. Sphalerite is abundant in the border zones of the district, in some instances being the predominating constituent of the vein filling, as in the Black Rock mine, where zinc ore is the chief product.

The less common minerals found in the copper veins are hübnerite, galena, barite, rhodochrosite, fluorite, and calcite. Passing from the copper to the silver veins, minerals characteristic of the silver vein area, rhodochrosite, galena, and barite become more and more abundant, the quartz and rhodochrosite finally forming the chief constituents of the gangue, although pyrite and sphalerite are present in notable amounts.

The great bulk of waste matter in the ore as mined and sent to the reduction works is altered granite, which forms from 50 to 70 per cent. by weight of the ore. The occurrence and association of the common minerals of the veins will be described in greater detail below.

Chalcocite.

Chalcocite is the most important copper mineral of the Butte district. As a common constituent of the copper ores it has been the source of not less than 60 per cent. of the total copper produced to date. It occurs in ore-producing veins of all ages and is abundant at all levels from the upper limit of the sulphides down to the greatest depths yet reached, extending, in many instances, more than 3,000 ft. below the surface.

Broadly speaking, the chalcocite of the Butte veins occurs in three

distinct forms : (a) as " sooty " glance, so-called, in which form it appears as a dull black coating on iron pyrite or other sulphides, or frequently developing a black amorphous powdery substance resembling " soot," where it has resulted from the replacement of the precipitant sulphide; (b) as massive steel-gray chalcocite ; and (c) in the form of crystals.   Of the three varieties named, the first two are of great economic importance.   Chalcocite crystals are of frequent occurrence but they have no commercial significance.

Sooty Chalcocite.—Sooty chalcocite is widely distributed as an ore-forming mineral, but it is confined particularly to those portions of the veins lying immediately below the zone of oxidation.   (See Plates II. and IV.)   This " sooty " glance zone, or belt, is the " chalcocitization zone " of Lindgren at Morenci.   In Butte it varies in vertical extent between wide limits, depending primarily on the mineralogical and physical character of the veins and the depth of the zone of oxidation. The veins of the Anaconda system, or the quartz-pyrite series, are deeply oxidized as a rule, and in these veins the sooty chalcocite reaches its greatest development.

In the fault veins the zone of oxidation is usually of but slight vertical extent, and since the primary ores seldom extend upward to the surface, the development of sooty chalcocite is of but little importance.   The barren portions of fault veins consisting of clay and crushed granite, occupying long stretches on the strike of the vein between ore shoots, do not offer an adequate source for copper to form sooty glance ores in quantity below the zone of oxidation.   The majority of the big chalcocite-enargite shoots of the fault veins do not extend upward to within 500 ft. of the lower limit of the oxidized zone. Where fault-vein ore shoots reach the surface they are found to be oxidized, and they are accompanied by the development of sooty chalcocite ores within the boundaries of the primary-ore shoot similar in every respect to the corresponding secondary enrichment belt of the quartz-pyrite veins.

In the veins of the Anaconda system, particularly in the great altered granite belts, sooty chalcocite is very important commercially.   It is found usually as a coating, or as a partial replacement of pyrite, and, less commonly, sphalerite, enargite, and chalcopyrite.   It replaces, completely or in part, the pyrite of the veins, also the stringers, veinlets, and fine disseminations of pyrite in the altered granite, not only within and along the veins, but in the intervening altered country rock. The chalcocitization of the altered pyritized granite lying between the more important veins has resulted in the development of a low-grade material similar in general character to the disseminated porphyry

ores of the Southwest, although in the Butte district such material is seldom rich enough to mine except when adjacent to the well-defined veins.

In the veins of the Anaconda, or quartz-pyrite, series (see Plates II. and IV.) the sooty chalcocite occupies a zone from 200 to 1,200 ft. in vertical extent, measured from the bottom of the zone of oxidation. The lower limit of this chalcocitization zone is ill-defined and extremely irregular, which is in marked contrast to the sharply defined lower limit of the zone of oxidation. The mineral is most abundant in the highest levels of the sulphide zone. As depth is gained it becomes less prominent, finally disappearing. On the accompanying maps, Plates II. and IV., the lines marking the lower limit of the sooty chalcocite zone are intended to mark approximately the elevation below which but little, if any, sooty chalcocite has been observed.

Massive Chalcocite.—Chalcocite in massive form is of wide distribution in the Butte veins, occurring in veins and masses of great purity. In color it is steel gray, exhibiting the usual conchoidal fracture. In the veins of the Anaconda system it is found in great abundance at all levels from the bottom of the oxidized zone to the greatest depth reached by underground workings. In the ore shoots of the Blue vein system chalcocite ores seldom extend upward to within 500 ft. of the surface. In the later Steward fault veins the rich chalcocite ore bodies were first encountered at a depth of from 1,000 to 1,200 ft.

As an ore mineral chalcocite is found filling fractures in quartz, pyrite, or other older vein minerals, and in irregular veinlets, seams, and stringers in altered granite within or along the veins. Chalcocite plays an important part in the formation of the great stock-work ore bodies of the Mountain View, West Colusa, and Leonard mines. In these properties the granite has been highly altered and much fissured. (See Plate I.) The shattering and alteration was followed by mineralization of the fissures and joint planes, chiefly by pyrite, quartz, chalcocite, and enargite, with occasional covellite. In the massive quartz-pyrite veins, such as the Anaconda and Syndicate veins, chalcocite is commonly found associated with quartz, pyrite, bornite, and enargite filling fractures or vugs in the earlier vein filling. Both in the older quartz-pyrite veins and in the later fault veins massive chalcocite is found in close association with pyrite, bornite, enargite, and quartz, forming an extremely complex mineral aggregate.

In the ore shoots of the Blue vein and Steward vein series, chalcocite frequently occurs as fine disseminations, seams, and splotchy masses mixed with iron pyrite, quartz, enargite, and bornite. It

commonly replaces the attrition clay and altered granite of these veins, often retaining the grooves, striations, and other markings characteristic of the original clay. Bodies of massive chalcocite are found in the ore shoots of No. 16 vein in the Rarus and Tramway mines. In these high-grade shoots, which are regarded as primary, enargite, pyrite, and quartz are also present intimately associated with the chalcocite. The massive chalcocite of the Blue and Steward veins and the major part of the chalcocite lying below the zone of sooty chalcocite in the Anaconda veins are believed to be of primary origin.

Enargite.

Enargite is of wide distribution both vertically and laterally in the Butte veins. It has been the source of from 25 to 40 per cent. of the copper output of the district. It is particularly abundant in the eastern portion of the district in the region of the Pennsylvania, Rarus, West Colusa, and Leonard mines. In the deep levels of the Gagnon and Original mines enargite is the predominating copper mineral and forms wonderfully rich and persistent ore bodies. Enargite is largely a product of a comparatively old mineralization period. It is found in copper-producing veins of all ages from the earliest up to and including the No. 16 fault veins of the Rarus and Tramway mines. Enargite is found in great abundance in the higher as well as in the lower levels of the Anaconda vein and other important veins of the oldest vein system, contrary to a former published statement [7] that it did not appear in the Anaconda vein higher than the 2,000-ft. level.

As a mineral of the veins, enargite usually occurs in interlocking aggregates of imperfectly outlined crystals in size from 0.25 to 1 in. across forming a solid mass, the individual growths always exhibiting characteristic cleavage surfaces. Beautifully formed crystals are frequently found from $\frac{1}{5}$ to 1 in. in length, but the larger sizes are usually less perfect than the smaller ones.

The more massive enargite is found occupying fractures in older quartz-pyrite veins, or replacing the altered granite of the vein in the form of veins and stringers and as splotchy masses along the cracks and joint planes of the granite. It is often intimately mixed with either pyrite, quartz, chalcocite, or bornite, and less commonly with covellite. Intimate mixtures of enargite, pyrite, and barite crystals contemporaneous in origin are frequently seen lining vein cavities.

Enargite is chiefly a product of primary ore deposition. In the

[7] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, pp. 77 and 107 (1912).

Shannon, Windlass, Enargite, South, and other veins of the eastern district it is found in great abundance in the upper mine levels, the greatest development occurring immediately below the lower limit of the oxidized zone and extending downward for from 400 to 600 ft. Chemical analyses show appreciable amounts of arsenic (see Table I. p. 38) in the oxidized portions of these rich enargite veins, but no marked development of chalcocite occurs in the sulphide zone immediately underlying these gossans. It seems not unreasonable under such conditions to assume that a part, at least, of the enargite is of secondary origin.

Bornite.

Bornite is one of the commonest of the copper sulphide minerals. It is almost universally present in the copper ores, associated with chalcocite, enargite, chalcopyrite, and other sulphides, but usually in subordinate amounts. As a source of copper it is especially important in the Steward and Original mines and in the mines of the North Butte section. The bornite occurrence in Butte is particularly interesting from the fact that while it seldom forms the chief copper mineral of the ore, it is rarely absent even in hand specimens.

In physical character the bornite of the Butte vein exhibits on fresh surface the characteristic horse-flesh color, tarnishing rapidly on exposure to a blue or green color. It is a frequent associate of chalcocite, chalcopyrite, enargite, and other sulphides.

Bornite occurs in copper veins of all ages at all levels from the oxidized zone to the greatest depths yet reached. It is largely a product of primary ore deposition along with enargite, chalcocite, and other associated primary minerals.

Chalcopyrite.

Although described in former years as the chief copper mineral of the primary ores of the Butte copper veins, chalcopyrite has proved to be of comparatively small importance as a source of copper. It is characteristically a mineral of the border zone of the central copper area. Chalcopyrite is practically unknown in the Anaconda, Mountain View, and other mines of the eastern part of the district. It is common in the extreme west end of the Gagnon mine, in the Lexington, West Gray Rock and in the veins of the Speculator mine. It is of frequent occurrence in the Elm Orlu mine, Alice, and other silver properties. Chalcopyrite is commonly a product of late vein-forming action, being frequently found in cracks, vugs, and cavities within older vein filling or as a thin coating or replacement of older sulphide minerals, particularly sphalerite, pyrite, enargite, and covellite. It has been noted in the form of small imperfect crystals in rare in-

stances, occurring in cracks in older vein filling and finely disseminated throughout the unaltered granite, particularly in the Altona and Amazon shafts in the eastern part of the district. The unusual occurrence in the Jessie vein of chalcopyrite-pyrite vein filling capping rich chalcocite-enargite ores at greater depths is of special interest and will be further described in the discussion of the formation of chalcocite. (See p. 100.)

In vertical distribution chalcopyrite is found at all levels. In certain localities, as in the Gagnon and Steward mines, it is more abundant in the deep levels than in the upper levels, in marked contrast to the chalcopyrite ores of the Jessie above mentioned.

Chalcopyrite in the Butte veins is largely a product of primary mineralization, although in certain instances it is believed to be possibly of secondary origin. Frequently it is the most recent mineral of the vein in which it is found, occurring as fine crystals in cracks or as a replacement of enargite, covellite, or chalcocite.

Covellite.

Covellite occurs as massive mineral and more rarely as crystals of the characteristic indigo-blue color in certain veins of the northern and eastern parts of the district. It was found in abundance in the Gray Rock and Edith May veins, particularly at comparatively shallow depths from the 700 to the 1,200 ft. levels. It occurs in considerable amounts in the Leonard vein from the 1,200 to the 2,000 ft. levels. The covellite of the Leonard vein frequently shows a partial alteration to chalcopyrite. An interesting occurrence of covellite was noted in the Skyrme vein of the High Ore mine, where on the 2,400-ft. level a large ore boulder broken open was found to be composed principally of covellite intimately associated with enargite, some bornite and large amounts of later chalcocite. This entire mass was in large part inclosed by a 0.5-in. covering of pyrite. At several points on the surface of the boulder the pyrite, enargite, chalcocite, and covellite alike were being altered and replaced by chalcopyrite as the latest mineral.

In the Butte veins covellite is believed to be largely of primary origin, if not entirely so. Its occurrence bears no relation to the surface or to the zone of oxidation, and the intimate association with enargite, bornite, and pyrite lends strong support to the primary view.

Tetrahedrite.

Tetrahedrite is unimportant as an ore of copper, although it has been found in small quantities in nearly every mine in Butte. It

varies in physical appearance from a dull gray to a bronzy metallic luster. The latter form occurs in considerable amounts in the Leonard mine, where it is remarkable for its high gold content. Lots of 50 tons have been known to run as high as $20 a ton. It is especially abundant in the stopes of the Gem vein above the 900-ft. level. The common variety has a characteristic cherry-red streak, which immediately distinguishes it from chalcocite.

Tennantite.

Tennantite is of frequent occurrence in the veins of the northeastern part of the district in the Badger, Speculator, and Gem mines, but is not of great importance as a source of copper. In massive form it is of a deep gray-black color, having a faint reddish tint. It is of more common occurrence as minute gray glistening crystals of a steel-gray color, filling or coating the walls of fractures in older vein filling. These crystals are believed to have formed during the alteration of enargite to chalcopyrite, marking possibly the regeneration of the mineral from the copper and arsenic lost in the transformation of enargite into chalcopyrite.

Sphalerite.

Sphalerite is a common mineral of the Butte veins. It is of wide distribution in all of the veins of the district except in that part of the copper-producing area extending easterly from the Never Sweat to the Leonard, Berkeley, and Silver Bow mines. (See Fig. 7.) Sphalerite is especially abundant in the Colorado, Gagnon, Poulin, Lexington, West Gray Rock, and Corra mines, and to the north in the veins of the old silver-producing area. Large bodies of sphalerite have been developed in the Elm Orlu and Black Rock mines, where it occurs chiefly in veins belonging to the Anaconda system. It is here found in unusual purity, the principal gangue minerals being quartz, rhodonite, and rhodochrosite. In marked contrast to the veins of the copper area, pyrite occurs but sparingly in the zinc ores of the Black Rock mine, running generally less than 2 per cent.

Sphalerite is found in abundance in veins and faults of the Anaconda, Blue, and Steward systems. It occurs at all levels from the oxidized zone to the greatest depths yet reached by mine workings. It is a primary vein mineral deposited contemporaneously with pyrite, quartz, galena, enargite, chalcocite, bornite, and other copper minerals, in veins of all ages. As is the habit with other sulphide minerals in the Butte veins, it is more abundant in higher levels in certain parts of the district, and relatively more abundant in deep levels of other localities. The relative proportion of sphalerite to copper content is

widely variable in different veins or at different sections within the same vein. In some instances veins have been found to contain an ore rich in sphalerite and low in copper in the higher levels with increasing proportions of copper as depth is gained. On the other hand, many veins with much zinc in the upper levels show no marked improvement in copper contents down to great depths.

Sphalerite is frequently found in intimate association with pyrite, bornite, chalcocite, galena, and quartz, and but rarely with enargite. Like enargite, sphalerite is often corroded or eaten away, the dissolving action being followed in many cases by the deposition of chalcopyrite, bornite, or chalcocite. The age relations of the sphalerite have not been worked out. It probably began to deposit at an early period of vein formation, and undoubtedly appeared in large quantities in certain parts of the Anaconda fissures long before the advent of the Blue system of fissures.

Galena.

Galena is found sparingly in the intermediate zone surrounding the central copper area. It is particularly abundant in the region immediately north of the Mountain Con mine in the Old Glory, Lexington, and Gray Rock veins. It has been noted in the Anaconda, Blue, and Steward fissures in the intermediate and peripheral zones. Galena is a common, but not necessarily plentiful, constituent of the manganese-silver veins and generally of the veins of the outer zones.

Silver.

Silver is universally present in the Butte veins. In the copper area silver is more plentiful in the veins carrying zinc. The enargite veins of the Rarus, Berkeley, and Silver Bow mines have the lowest silver content of any in the district. Silver increases toward the border area, where the proportionate amounts of bornite, chalcocite, and sphalerite become greater. The mineralogical nature of the silver occurrence has not been definitely determined. It is frequently seen in native form associated with chalcocite and bornite. Argentite has been noted in the veins of the silver area.

Gangue Minerals.

Quartz.—Quartz is the most abundant vein mineral of the district. It forms the chief constituent of the gangue of the copper and zinc veins, occurring in both massive and crystal form. Quartz is a product of vein-forming activity of all ages from the earliest known veins following the appearance of the quartz-porphyry dikes up to the close of the primary ore-forming period marked by the Steward-vein ores.

Several generations of quartz are readily observed in the vein filling of Anaconda veins, all of which are of primary origin. Secondary quartz as a product of meteoric water circulation has not been observed, excepting possibly within or near the lower limit of the zone of oxidation. Quartz probably forms 70 per cent. by volume of the vein filling of the Butte veins.

Pyrite.

Pyrite is the most widely distributed of the sulphide minerals. It occurs both massive and crystalline in veins and fissures of the Anaconda, Blue, and Steward systems. Pyrite occurs sparingly as a primary constituent of the Butte granite, and abundantly in a finely disseminated condition in altered granite resulting from the attack of thermal waters upon the dark silicates of the original rock. In the veins of the district pyrite is more abundant in the early copper veins belonging to the Anaconda system than in later fault veins, and much more abundant in the veins of the central copper area than in the manganese-silver veins of the border areas. In vertical distribution it is doubtful if there has been a measurable change in the total pyrite present between the higher and the deep levels of the mines. In the regions of sooty chalcocite pyrite has been to some extent replaced by secondary chalcocite. Below the zone of sooty chalcocite no marked change has been noted.

Manganese Minerals.

Rhodonite and rhodochrosite are found in great quantities in the veins of the peripheral zone surrounding the copper-producing area of the district, but rarely in the important copper-producing veins. These minerals are also abundantly associated with the sphalerite of the Black Rock veins.

Other Minerals.

Hübnerite, fluorite, barite, and calcite are of frequent occurrence in the copper area. *Hübnerite* has been noted in the Steward, Leonard, Mountain View, and many other mines, where it appears as one of the earliest-formed minerals. It is found in considerable quantities in certain veins, 2 miles east of the Pittsmont mine. In the Mountain View mine blade-like enargite after hübnerite has been noted from the High Ore vein, a northwest-southeast fault of the Blue system.

Barite in characteristic tabular crystals is occasionally seen projecting into cavities in the veins. It is pale brown in color. Excellent crystals have been found in the Parrot mine.

Calcite is rarely seen in the copper veins, but it is commonly formed in veinlets along joint planes of the granite in areas of fractured or crushed character, where alteration processes have but slightly affected the rock. It is unknown in areas or zones of altered granite. Where found in copper veins it has always proved to be the most recent mineral, often filling cracks through chalcocite, chalcopyrite, and other copper minerals.

Fluorite has been found in the Parrot vein, in the Blue and Steward veins, and in considerable quantities in the Black Rock vein on the 1,200-ft. level of the Black Rock mine. In the Parrot mine on the 1,200-ft. level fluorite and enargite were intimately associated and apparently contemporaneous. Later chalcopyrite was observed replacing the enargite.

THE ORE DEPOSITS.

The ore deposits of Butte are essentially of the fissure-vein type. They have resulted from the mineralization of fissures accompanied by replacement of the country rock. The fissure systems, as previously described, belong to at least six, or possibly seven, distinct periods of fracturing, and many examples of extreme complexity are presented. The oldest or first-formed fractures, composing the Anaconda system, have been continuously mineralized and are remarkably free from sudden changes in vein filling, a feature which presents a marked contrast to the later mineralized fault fissures of the Blue and Steward systems. In the latter the ore occurs in great lenses, or shoots, with intervening stretches of barren vein characterized by crushed country rock and fault gouge. The fissure-vein structure is the rule throughout the district. The ore-bodies display rather well-defined boundaries, when broadly considered. Important exceptions, however, are found in the Leonard, West Colusa, Rarus, and Tramway mines, where the largest ore bodies are more in the nature of mineralized highly fissured granite, having boundaries which are often commercial rather than geological The mineralization of the early complex fracture systems, followed by later faulting, has resulted in a most complicated arrangement of ore bodies. Reference to Plates I. and II. will convey to the reader a more comprehensive understanding of these structural relations than can be conveyed by detailed description.

The valuable metal content of the ores is chiefly copper, with subordinate but important amounts of silver, gold, and zinc. As mined, 60 to 80 per cent. by weight of the ore is altered granite, which usually, though not always, carries sufficient quantities of valuable minerals in seams, impregnations, or disseminations to constitute ore.

Irregularities in the vein boundaries, horses, pinches, and included granite, necessitate stoping widths often in excess of the actual thickness of ore streaks. The copper ores invariably carry commercially important quantities of silver. The typical manganese-silver ores contain only traces of copper; and the newly developed zinc ores of Elm Orlu and Black Rock mines carry considerable silver, but rarely appreciable amounts of copper.

Distribution of Ore Types.

An interesting geological condition is found in the unmistakable concentric zonal arrangement of certain ore types, based on mineral composition, around a central copper zone. It was observed in the
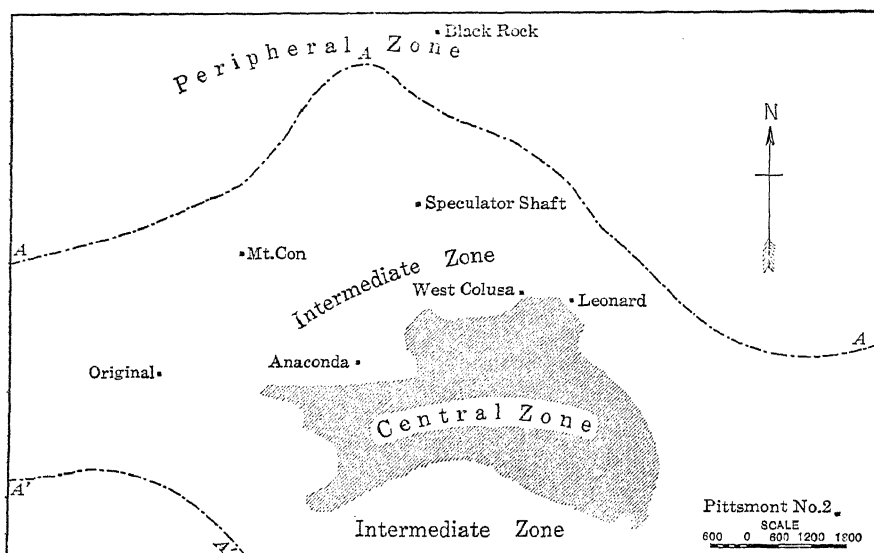
FIG. 7.—PLAN MAP AT ELEVATION 4,600 FT. ILLUSTRATING GENERAL DISTRIBUTION OF ORE TYPES WITH REFERENCE TO THE CENTRAL COPPER ZONE.

early days of mining that ores from different mines exhibited considerable variation in mineralogical composition, but it has been only in the more recent years that underground developments have progressed to such an extent that the apparent orderly arrangement of the various types can be approximately outlined. From the information now available, the writer has formulated certain generalizations concerning these features, and on the accompanying map, Fig. 7, an attempt has been made to indicate as accurately as possible the zones or belts which are typified by characteristic mineral associations.

In brief, these zones may be grouped as follows:

1. A main or central copper zone occupying largely the grea   area

of altered granite in the vicinity of the Mountain View mine, in which the ores are characteristically free from sphalerite and manganese minerals. This zone is represented by the shaded area in Fig. 7.

2. An indeterminate zone of irregular width nearly surrounding the central copper zone, in which the ores are predominantly copper, but are seldom free from the mineral sphalerite, and near the outward boundaries, $A-A-A$ and $A'-A'$, the manganese minerals rhodonite and rhodochrosite are of frequent occurrence.

3. An outer or peripheral zone of undetermined width bordering the intermediate zone, in which copper has not been found in commercial quantities. The vein filling is chiefly quartz, rhodonite, sphalerite, pyrite, and rhodochrosite. In this zone are included the manganese-silver veins of the Alice, Moulton, and Magna Charta mines on the north and the Emma, Ophir, Travonia, etc., on the south.

It should be kept in mind that the dividing lines between these three zones shown on the map are from necessity of an arbitrary nature. The passing of one zone into another is gradual, and cannot, as a matter of fact, be correctly represented by a mere line. Underground developments will never be of sufficient extent to permit of this extreme refinement.

The Central Copper Zone.—Within the central zone represented by the shaded area in Fig. 7 are found the typical copper ores in which the copper minerals are predominantly chalcocite and enargite in a gangue of pyrite and quartz. Bornite is also present, but in proportionately small amounts. Covellite is uncommon, having been found only in the Leonard and Mountain View mines associated with chalcocite, enargite, and later chalcopyrite. Chalcopyrite and sphalerite are extremely rare, having been noted in but few instances. The manganese minerals rhodonite and rhodochrosite are unknown, as is galena. Silver is a universal constituent of the veins of this area, but in less quantity than in the ores of the intermediate zone. The ratio between the silver and the copper is approximately $\frac{1}{3}$ oz. of silver to per cent. of copper.

It is of special interest to note that the central copper zone coincides in part with the great zone of rock alteration, and again a feature of interest is found in the fact that within the area there is no essential difference in mineralogical composition of the vein filling in veins of different ages, although there may be great variations in the relative amounts present. In one locality a vein of the Anaconda system may contain proportionately more enargite than a nearby vein of the Blue system, while in other localities the reverse is true. As a general con-

dition, however, it is believed that the later veins of the Blue system have more chalcocite and bornite, and less enargite proportionately than the veins of the Anaconda system. In the ores of the Steward veins the chalcocite ratio is especially high.

The Intermediate Zone.—In the intermediate zone surrounding the central copper area there is a noticeable change in the mineralogical composition of the veins. This change consists chiefly in the addition of the mineral sphalerite to the general type of vein filling of the copper zone above described. Outwardly from the central copper area the sphalerite does not appear suddenly in great quantity, but it comes in rather gradually. This feature is variable in different localities; for example, in the Moonlight mine there is but little sphalerite in the veins of the east end of the mine, but westerly even in the same veins, within a distance along the strike of 1,000 ft., the ores become very rich in zinc. In the Anaconda vein westerly from the Never Sweat mine sphalerite increases in amount slowly and gradually, and unusual quantities of zinc are found only at distances of 3,000 ft or more from the general outside limit of the central copper zone.

In addition to the changes noted in zinc content, other mineral variations take place that are of considerable interest. The manganese minerals rhodonite and rhodochrosite begin to appear in small quantities toward the borders of the intermediate zone, and increase perceptibly toward its outer limits. Among the copper minerals, as compared with the central copper zone, there is less enargite in proportion to the total copper present, but proportionately greater quantities of bornite, chalcopyrite, tetrahedrite, and tennantite. Chalcocite apparently remains about the same relative to the total copper present as in the central copper zone. The silver content increases materially, the ratio to copper content being 1 oz. of silver to 1 per cent. of copper as a general average. It should be remembered that the facts above outlined are generalized conditions and that locally extreme variations occur. The net result, however, is a decrease in copper content with an increase in silver, zinc, lead, and manganese. Quartz as a gangue mineral does not vary perceptibly, but pyrite is less abundant toward the outside limits of the intermediate zone than in the central copper zone and certainly much less common in the peripheral zone than in the intermediate and central zones. In the intermediate zone the veins of different systems carry the same variety of minerals, but in widely different proportions. The veins are more zincky in some localities than in others. Those of the Anaconda system are exceptionally high in pyrite and quartz in certain localities, and at

other places sphalerite is unusually abundant. There are certain areas in the Mountain Con and Diamond mines where the ores closely resemble those of the central copper zone, and, curiously enough, these areas are closely associated with the earliest vein development which had attended intense rock alteration.

The Peripheral Zone.—As with the central and intermediate zones, the line of division between the intermediate and the outer zones is largely an arbitrary one. It marks the general outside limit of present known commercial copper deposits, and therefore it is in nowise intended to be construed as an attempt to mark the possible limit of copper ores. Up to the present time the ores of this zone have been valuable principally for silver, gold, and zinc. The vein filling is characterized by the abundance of the manganese minerals rhodochrosite and rhodonite. Sphalerite is present in great abundance, forming in some instances ore bodies of value. Copper is sparingly present, chiefly as chalcopyrite, tetrahedrite, tennantite, and rarely chalcocite and bornite. Pyrite is common, but in relatively much less quantities than in the two zones above described. Quartz is the most abundant gangue mineral. Galena is also present in considerable quantities intimately associated with sphalerite and of the same age. The width of this zone is indefinite and irregular and there is a noticeable change in the mineralogical character of the vein filling at greater distances from the central copper zone. Extending outward from the peripheral zone the fissures appear to become less mineralized; the manganese, pyrite, zinc, and other sulphides largely disappear, the vein filling consisting principally of quartz with scattered pyrite; and, curiously enough, arsenopyrite is noted at extreme distances, 2 miles or more from the central copper zone.

The zonal arrangement of ore types above described is repeated on a smaller scale in certain ore shoots of the Blue veins in the northeastern part of the district. In the shoots of the Jessie and Edith May veins, for example, the mineral composition differs in the central part of the shoot from that at the ends and along its walls. High-grade chalcocite-enargite-bornite ores form the central part of the ore shoot with but little sphalerite and chalcopyrite and shade almost imperceptibly into zincky ore with chalcopyrite toward the ends of the shoots.

Vein Systems.

Of the seven distinct periods of fissuring in the district only three are known to be ore producing: namely, the Anaconda system, the Blue system, and the Steward system. The Anaconda system is the

most extensively mineralized, the Blue system is less so, but is extremely important as an ore producer, while the Steward fissures seldom contain workable ore bodies, although in many instances they appear to have exerted an important influence on older veins. The mineral composition of the ores of the various vein systems is of marked similarity. A suite of hand specimens typifying the ores of the Anaconda veins does not differ materially from a similar suite collected from the later fault veins. The distinctions where recognizable are in the relative quantities of minerals present, or in matters pertaining to physical character. The variation in mineral composition of the veins is a matter of geographic position, rather than of geologic age as above outlined.

The Anaconda Vein System.

The Anaconda system of veins is the oldest geologically and the most important commercially of any in the district. It comprises a great series of veins having a general east and west strike, which have resulted from the mineralization of the Anaconda system of fractures previously described. The largest and most important of these veins is the Anaconda, variously called the Gagnon, Original, Parrot, Anaconda, Mountain View South, etc., named from the many adjoining shafts through which it has been worked. Another important member of this system is the Syndicate vein, or lode, of the Mountain Con mine, one of the big producers. Other highly productive veins belonging to the Anaconda system are the O'Neill, Moonlight, Berkeley, Windlass, Shannon-Colusa, Modoc, Bell-Speculator, Middle, West Gray Rock, Chief Joseph, Badger, and many others of less importance, but having the same general geological characteristics. It is believed that the silver veins of the Alice, Moulton, Lexington, and other mines of the north district are also mineralized fractures belonging primarily to the Anaconda system. In mineralogical character, however, they differ materially from the copper veins above named.

Structure.—The veins of the Anaconda system are remarkably continuous. The ore seldom shows sudden changes in width or general mineralogical character. Many of the important members of this system have been stoped continuously on strike for thousands of feet. As a replacement of relatively firm rock the ore is usually hard and massive, at times exhibiting imperfect banding due to replacement of sheared granite, or deposition of mineral along closely spaced fractures.

The veins often split or branch, both on strike and dip. Where

continuously stoped they are frequently composed of two or more closely spaced ore streaks, called the "foot-wall streak" and the "hanging-wall streak." It is not unusual to find a stope face showing four or five roughly parallel ore streaks varying in width from an inch up to several feet. The included granite and wall rock are highly altered and commonly netted with smaller seams of mineral similar in a general way to the large ore streaks. The variable mineralogical nature of the many ore seams, veinlets, etc., is a notable feature not only of the veins of the Anaconda system, but of all of the veins in the district. For example, a drift face may show a well-defined foot-wall streak 2 ft. wide composed almost entirely of quartz and pyrite assaying less than 2 per cent. of copper. The hanging-wall vein, separated from the foot-wall streak by perhaps 2 ft. of highly altered granite, may be very rich, consisting largely of chalcocite, enargite, bornite, and pyrite, assaying 25 per cent. of copper. However, as stopes are made above the drift, the rich streak may show only enargite and pyrite or only chalcocite, while the lean foot-wall vein in the stopes may become richer by the appearance of enargite, chalcocite, or bornite. These extreme local variations in veinlets and ore streaks are extraordinarily common to veins of all systems, although the general average valuable metal content of the ore as mined in a particular locality may not show much general longitudinal or vertical variation over hundreds of feet. It is a notable fact that while the variety of minerals present in a given vein seldom suffers marked changes within short distances, the relative proportions of minerals reach extremes.

The component ore or vein streaks may or may not be included within fairly well-defined boundaries. Usually one of the ore bands is wider, more prominent, and more persistent than the rest, the smaller streaks maintaining a rough parallelism separated by vein granite from the main fissure. The most important variation from the common type of vein above described is found in the Shannon-Colusa vein. The stoping width of the Anaconda veins varies from 5 up to 100 ft., with a general average of 20 ft. The Anaconda and Syndicate veins average 30 ft. or more. The stopes of the Leonard mine within the big stock-work ore bodies often run as high as 400 ft. from north to south. Within this extreme width, however, there are many bands or ribs of waste, and as a rule the actual stope is not solid ore, from 10 to 30 per cent. of the ground broken being sorted out and rejected. Among the less important veins 10 ft. is a common stoping width. In most cases the actual vein filling represents only

a fractional width of the stoped material, the remainder being mineral-bearing altered granite.

Minerals of the Anaconda Veins.—The veins of the Anaconda system carry more quartz and pyrite than the later veins, though the term "quartz-pyrite," formerly much used to designate the veins of this system, is, in the strict sense, a misnomer. Quartz and pyrite are present in great quantities in the Blue veins and form the chief gangue minerals, but in less quantities in proportion to the copper minerals present than in the earlier veins. The greater continuity and uniformity of the hard vein filling, its freedom from fault clay and crushed granite, together with the proportionately greater amounts of quartz and pyrite, gave rise to the early usage of the name "quartz-pyrite" veins as distinguished from the typical fault fissure not known to contain ore bodies. That "quartz-pyrite vein system " is not, strictly speaking, a proper designation, but often misleading, may also be readily understood by a casual inspection of representative ores from the different veins of the district.

The general mineralogical character may be best illustrated by a more detailed description of some of the more prominent veins belonging to the Anaconda system. Of those described below the Anaconda is typical of the important well-defined continuous compound fissures. The Shannon-Colusa illustrates fissuring of remarkable complexity and ore bodies of an unusual type. The Syndicate vein illustrates certain structural features of interest, while the Gray Rock vein is of interest as a copper vein well toward the border of the copper-producing area.

The Anaconda Vein.—Beginning in the Gagnon mine on the extreme west, the principal copper minerals are chalcocite, enargite, bornite, and chalcopyrite, named in order of their abundance. The gangue minerals are quartz, sphalerite, and pyrite, with occasional occurrences of rhodochrosite and hübnerite. In vertical arrangement there is possibly more chalcocite and less enargite and bornite in the upper levels than in the lower levels of the Gagnon mine. The sphalerite is only slightly variable as between the higher and lower levels, probably less in the latter. The most notable change in depth is the remarkable development of enargite in the bottom levels, accompanied by unusual quantities of later chalcopyrite, largely a replacement of enargite. In the Gagnon mine the silver tenor of the ore has decreased in depth. The oxidized zone here is from 100 to 200 ft. deep, accompanied by a considerable development of sooty or secondary chalcocite whose vertical extent is in the neighborhood of 400 ft.

Easterly from the Gagnon mine, in the Original and Steward, the oxidized zone with the accompanying zone of sooty chalcocite shows no marked change from that just described. Of the copper minerals, enargite and chalcopyrite decrease, while chalcocite and bornite increase in amount. Quartz and pyrite increase, especially the latter, but sphalerite shows a marked decrease. The silver content is slightly lower than further west. Rhodochrosite and fluorite have been noted in the Steward and Parrot mines.

Taking the next step eastward into the Never Sweat, Anaconda, and St. Lawrence mines, the enargite and bornite have continued to increase slightly, chalcocite increases, silver content becomes lower, sphalerite and chalcopyrite disappear completely, while on the whole, the vein becomes much more massive. The gangue, instead of being granitic, is hard, composed chiefly of quartz and pyrite. Hübnerite is noted occasionally; rhodochrosite and fluorite are unknown. In the Never Sweat-St. Lawrence section the oxidized zone is of unusual depth, averaging 300 ft. Below, the secondary chalcocite zone shows extensive development, having a vertical extent of from 600 to 800 ft.

Easterly from the St. Lawrence the Anaconda is faulted north by the High Ore vein into the Mountain View mine, where the mineralogical character does not differ materially from that shown in the Never Sweat-St. Lawrence region. The copper minerals are enargite and chalcocite, with unusually small amounts of bornite in a quartz pyrite gangue. Southeasterly, where the main Anaconda vein apparently forks and in a large measure loses its identity, there is no noticeable change in mineral content, except as to relative quantities present, until the region of the Pittsmont or East Butte property is reached, where the only change is in the reappearance of sphalerite in considerable quantities. Chalcopyrite does not again appear except in rare cases, chalcocite and enargite forming the chief source of the copper. The silver content decreases slightly southeasterly from the Mountain View. The rarer minerals noted are fluorite in the Parrot mine, barite occasionally at different points in the vein, and hübnerite in the Mountain View mine.

On the whole, the veins of the Mountain View, Rarus, and Pennsylvania mines have more enargite but very much less bornite in proportion to the total copper present, than those of the Steward, Anaconda, Moonlight, and St. Lawrence mines.

The Shannon-Colusa Vein.—The Shannon-Colusa vein is a strongly mineralized fissure forming the north boundary of the complex Anaconda fracture zone previously described. In the Mountain View mine it has been developed as a compound fissure striking

slightly north of east, composed usually of a single vein in the upper levels, but of two or more branches at greater depths. The vein dips from 80° to 85° north. There are, occasionally, nearly north and south veins branching toward the south, but the north wall is generally clean cut and well defined. This comparatively simple vein structure is maintained eastward into the West Colusa mine, where it suddenly becomes exceedingly complex. Great mineralized fissure zones forming remarkable ore bodies break suddenly to the south nearly at right angles to the main foot-wall fissure. The granite between and forming the country rock of the complex fissure aggregates is richly mineralized with chalcocite and enargite as disseminations or veinlets following the joint planes of the granite.

In the Leonard mine the development of the north-south vein structure does not appear higher than the 600-ft. level. Figs. 1 and 2 illustrate the relation between the veins of the Leonard 300-ft. level and the fracture complex of the 1,200-ft. level. On the 300 there are four well-defined veins having approximate east-west strikes. The Colusa, the most northerly one, is a large well-defined vein, averaging 40 ft. in width, having an 85° dip to the south. At a depth of 600 ft. it experiences a sudden change to a flatter dip, due in part to faulting by the Rarus fault, causing it to appear on the 700-ft. level, 320 ft. south of its position on the 600-ft. level. From the 700-ft. level downward it meets the vertically dipping Gambetta veins, and it is below this level that the development of the north-south fissuring begins. The north-south veins are often well defined and carry wonderfully rich bodies of chalcocite-enargite ore, with occasional admixture of covellite associated with later chalcopyrite.

The Gambetta veins are intersected between the 300-ft. and 500-ft. levels and are displaced northward by the Rarus fault. Above the fault they carry but little commercial ore except near the Gambetta shaft. Below they are notably rich in enargite, often showing from 2 to 5 ft. of nearly pure enargite, with numerous veinlets of coarse enargite penetrating the vein walls. Where intersected by a crosscut on the 800-ft. level the Gambetta vein showed more than 6 ft. of solid, coarsely crystalline enargite, with little or no chalcocite, quartz, or pyrite. The oxidized zone of the Gambetta veins is slight, being not more than 60 ft. and usually much less. There is but a slight development of sooty chalcocite below; in fact, not enough to make the vein filling of commercial grade. The zone of oxidation in the Colusa vein, which varies from 100 to 150 ft. in depth, is accompanied by a marked enrichment of the veins below by sooty chalcocite. This enrichment zone, however, barely reaches the 600-ft. level. In the big

flat stopes connecting the 700-ft. and 600-ft. levels and in the big ore zones of deeper levels, the valuable copper minerals are chalcocite and enargite, with occasional bunches of covellite associated with chalcopyrite.

The Syndicate Vein.—The Syndicate vein of the Mountain Con group of mines is a massive quartz-pyrite vein with a curving strike from N. 70° E. in the Buffalo mine to a more easterly and southeasterly strike eastward from the Mountain Con mine. (See Plate I.) In general habit it does not differ materially from the main Anaconda vein above described. There are the common splits, or branches, on strike and dip, including horses of granite. The granite of the walls is highly altered and contains the usual minor mineralized fissures roughly parallel to the main vein. The Syndicate is cut and displaced by the Nappa, Dernier, Midnight, and other fault veins belonging to the Blue system. The Bell, a strong northeast fault, crosses the Syndicate vein at an acute angle, at times following it as a strike fault for several hundred feet.

In mineralogical character there is a notable difference between the Syndicate and the main Anaconda vein. The former is more massive, with a larger proportion of pyrite as a gangue mineral. There is more bornite and less enargite in proportion to total copper present than in the Anaconda. Chalcocite is abundant. There is the usual oxidized zone of from 150 to 250 ft. thick, accompanied by important secondary enrichment below. There is less chalcopyrite than in the Gagnon mine, but more than in the neighborhood of the Mountain View mine. The Syndicate vein contains large amounts of sphalerite in the Poulin, slightly less in the Buffalo, Mountain Con, and eastward. In the immediate vicinity of the Mountain Con shaft for a short distance on strike sphalerite is almost entirely absent. There is a tendency on the west to develop minerals characteristic of the manganese-silver vein area to the north. Barite, galena, tetrahedrite, and tennantite are frequently seen in the ores of the Poulin mine. The Old Glory vein of the Buffalo carries unusual amounts of galena.

West Gray Rock Vein.—The West Gray Rock vein has been worked only in the Gray Rock mine. It is a well-defined vein of the Anaconda system, exhibiting the usual characteristics as to general strike, continuity of mineralization, etc. It appears to split into several branches, dying out quickly both east and west. It is a typical hard quartz-pyrite vein having more quartz and less pyrite than the Syndicate. Sphalerite is always present. Galena, chalcopyrite, and barite are occasionally noted. Chalcocite and bornite are the chief copper

minerals. Engarite is less important that in most of the copper veins.

It is faulted by the High Ore, the South Bell of the Blue system, and the La Plata of the Steward system, and the Corra fault, a north-dipping fissure possibly of the same age as the Rarus.

Physical Changes Effected in Anaconda Veins by Faults.—Where in contact with the intersecting fissures of the Blue vein system the Anaconda veins are usually shattered and broken, often exhibiting step faulting with a disturbance of orientation of the vein as the fault is approached. There is frequent strike faulting locally along the intersected vein, due to the general disturbance of the hanging- and foot-wall country-rock near the fault. Blue vein fault fissures often split where passing through older veins and develop the phenomena of step faulting.

The physical changes caused by faults meeting the Anaconda veins at acute angles or strike faults is more marked than that caused by faults like the Blue and Rarus crossing at obtuse angles. With strike faults along other veins, of which the Gagnon, Moonlight, and Bell-Speculator are notable examples, the movement may take place along the foot or the hanging wall of the vein or both. If the movement is extensive the older vein filling becomes more or less broken. If ore-bearing solutions are introduced by the later faulting, or if ore-forming waters are traversing the older vein, the new fractures and cracks in the old vein filling may become filled with mineral again forming in solid veins. This process of breaking and re-cementing of older veins by new primary ores is a common feature of the Butte veins, and where the process is many times repeated the chronological sequence of mineral formation cannot be satisfactorily determined.

Often where a strike fault crosses older vein filling diagonally from hanging-wall to foot-wall, the displacement along the fault causes the two segments of the vein to become separated on strike by a barren stretch of the fault fissure, giving the general effect of separate ore shoots, a feature early described by R. G. Brown.[8]

Mineralogical Changes Due to Appearance of Faults.—The changes in mineral composition of the Anaconda veins effected by later fault influence cannot be fully determined. It is certain that the Anaconda veins were well formed and existed as solid quartz-pyrite veins containing unknown quantities of copper minerals at the time the Blue fault fissures appeared, as shown by included blocks of drag ore found at intersections with Blue veins. The extensive mineralization of the

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[8] Brown, R. G., The Ore-Deposits of Butte City, *Trans.*, xxiv., 555 (1894).

Blue fissures, identical in character with that of the Anaconda veins, indicates that there was no substantial period of quiescence separating the two vein series. With an overlapping of these periods, marked possibly by a slight change only in the character of minerals deposited, the problem concerning the influence of the Blue veins on earlier-formed veins becomes a complex one and difficult of solution.

Certain facts have been observed which tend to throw some light on this question. It is found that the crossing of older veins by fault veins of the Blue and Steward is not always accompanied by un-doubted enrichment of the former. In fact, the stope and assay records show that as a general rule there has been no marked local enrichment at these intersections. There are many exceptions, however, where, at certain elevations along the line of intersection, between veins of the Anaconda system and later fault veins, both the old vein and the later fault fissure contain unusually rich ores. Careful observations of a number of such examples of enrichment show that they mark not merely the intersection of the two fissures, but the intersection of the older vein and an ore shoot lying within the plane of the fault fissure. The enrichment, therefore, bears no relation to the surface or secondary influence, but it occurs where the ore shoot happens to cross the older vein segments. A good example is found in the Parrot mine where a Blue vein ore shoot meets the Parrot vein between the 1,200-ft. and the 1,500-ft. levels.

It has been suggested that the Blue veins have been prominent zinc ore channels and that the Black Rock and other sphalerite veins probably of Anaconda age, of the north district, owe their unusual zinc content to the Blue vein fissures. The Black Rock vein appears to be of Anaconda vein age, as it is cut and displaced by northwest fissures of the Blue system, and no doubt its mineral content has been influenced by these intersecting veins. But it might be said with just as good reason that since these same northwest faults of the Blue system contain copper ore farther south, they were responsible for the copper in the earliest copper veins. The fact is, the Blue veins in the zinc regions contain sphalerite, quartz, and manganese minerals just the same as the other veins of that particular section. In the central copper area, however, the Blue veins carry no sphalerite, neither do the Anaconda or Steward veins; they all contain the ore and gangue minerals common to that locality. Intermediate between the copper and zinc areas, the Blue veins, which extend directly and continuously from one area to the other, contain ores of intermediate mineralogical composition, getting more zincky with less copper toward the zinc area, and less zincky with more copper toward the

copper area. Finally toward the north the copper minerals fail with increased sphalerite, manganese, and quartz, and less pyrite, while toward the copper area, sphalerite and the manganese minerals drop out completely, while pyrite and the copper minerals, chalcocite, enargite, and bornite, increase.

The above general statements apply with equal force to the fault-veins of the Steward system or any faults or fissures which have served as channels for uprising primary ore solutions. The Steward veins, like the Blue, exhibit the same variations in mineralogical composition which are believed to be functions of geographical position. (See pp. 58–61, inclusive, Distribution of Ore Types.) The direct effect of the Steward faults on the mineral composition of the Anaconda veins is a more obscure problem than in the case of the Blue veins, because the Steward fissures are so scantily mineralized that, even where in direct contact with older veins for long distances on strike, there are logical reasons for doubting that any primary enrichment whatsoever has resulted directly from these faults. In certain instances, however, it is believed that older veins have been enormously enriched by faults of the Steward system. Important examples may be cited, such as the Moonlight vein in the Anaconda mine, the Gagnon vein in the Original and Gagnon mines, the Bell-Speculator, and others. The No. 16 vein in the Rarus and Tramway mines contains remarkably rich primary chalcocite ore where it possibly acted in part as a feeder, exercising a marked influence on the mineral content of the big ore bodies of the Leonard, Tramway, and Rarus mines.

Since fault-vein ore occurs in separated shoots along the strike of the fissures, it is not improbable in the case of strike faulting that the crackled and reopened older vein filling has supplied important new channels connecting the widely separated ore channels of the fault veins, in which case the enriching influence upon the old vein filling might be very great.

Influence of Faults on Secondary Enrichment.—The influence of faulting on the secondary enrichment processes which have been active in the Anaconda veins is not easily determined. The breaking and shattering, by faults, of the old massive vein filling has undoubtedly permitted a more rapid and extensive oxidation of the vein and a correspondingly greater enrichment below. As in the case of primary minerals, the strike faults have exerted more influence on the older veins than the cross faults; that is, where the coincidence of the old vein and the strike fault occurs in levels relatively near the surface. In the St. Lawrence and Mountain View mines secondary enrichment

has reached unusual depths, owing to intense fissuring and a highly altered and mineralizing condition of the granite, and to faulting of Anaconda veins.

Weed's statement[9] that the early quartz-pyrite veins belonging to the Anaconda system were too low grade and unworkable except where fractured and enriched through the action of descending sulphate waters, has not been corroborated by the writer's own observations. The assertion that workable ores in the veins of the Anaconda system are localized and occur only at or near intersections with later fissures is entirely disproved by mining operations.

The Rarus fault has not influenced the mineral content of the ores of older veins, either in regions near the surface or at greater depths. The segments or blocks of ore included within the walls of the Rarus fault are no richer than the original vein. There has been no primary enrichment traceable to Rarus fault influence and it is believed that this fault has exerted but slight influence even in the case of the sooty chalcocite enrichment.

Vertical Distribution of Ore Minerals in Veins of the Anaconda System.—The upper parts of the Anaconda veins are oxidized to depths varying from 50 to 400 ft. The mineralogical composition of the oxidized portions of the veins is fully described under the subject of superficial vein alteration on preceding pages of this paper. Without exception in the copper district there is a development of sooty chalcocite immediately below the zone of oxidation in the Anaconda veins, which is found to be of variable vertical extent, depending primarily upon many factors, such as depth of the zone of oxidation, physical character of the vein, copper content of the primary ore, etc. Since the Anaconda veins were, in many instances, rich with primary copper minerals such as enargite, bornite, and chalcocite, apparently before being subjected to secondary influences of importance, the addition of a large amount of copper in the form of secondary chalcocite resulted in marvelously rich ores extending many hundreds of feet below the zone of oxidation. In the large quartz-pyrite veins, such as the Anaconda, Colusa, and Syndicate, the wonderful bonanzas of the upper levels were examples of the secondary enrichment of already rich primary vein filling. Below the zone of secondary enrichment there are no great variations in the mineralogical composition of the veins on the dip at any particular vertical section taken through the vein, but no general rule holds good for all veins. For

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[9] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, p. 95 (1912).

example, in certain veins, such as the Johnstown, Shannon, Enargite, Windlass, and many others of the Mountain View mine, the predominant copper mineral in the higher levels is enargite, while in the Gagnon and possibly numerous other veins enargite is much more abundant in the deep levels than near the surface. Analogous conditions hold in the case of primary chalcocite, bornite, and sphalerite. In the "stock-work" ore bodies of the Leonard mine primary chalcocite is vastly more abundant at great depths than above the 800-ft. level. The same is true of the Moonlight, Original, and other veins.

Concerning the presence of the gangue minerals, quartz and pyrite, much difficulty is encountered in an endeavor to arrive at definite conclusions as to increase or decrease with depths. As indicated by the uneven distribution of these minerals along the veins, it is not improbable that they are principally deposited in irregular zones or shoots within the plane of the vein, so that observation on a single vertical cross-section through a vein does not accurately determine the general vertical distribution of either mineral. As a general observation on this point, however, it is believed by the writer that proportionately to the total gangue present, there is more quartz and less pyrite in the higher levels in the Anaconda veins than at great depths. Not enough data have been collected, however, from which reliable deductions can be made.

In the manganese-silver veins belonging to the Anaconda system but little information can be had on the matter of vertical distribution of minerals, owing to the inaccessibility of the mine workings.

Blue Vein System.

The veins belonging to this system have resulted from the mineralization of the Blue fissure system previously described. (See pp. 18–20.) The veins known to be of greater or less importance as ore producers are the No. 1, Clear Grit, Blue, Diamond South, High Ore, South Bell, Skyrme, Adirondack, Edith May, Jessie, Gem, and Crœsus. In the earlier days of copper mining in Butte the presence of these well-defined copper-bearing fault veins was not recognized. Even for a long time after they were known as faults, their importance as ore carriers was not known. Their tardy development was due to two facts, later determined: (a) the ore bodies rarely reach the surface so as to come within the observation of the prospector, and (b) the early copper vein developments were along quartz-pyrite veins of the Anaconda system, little attention being paid to cross fissures, known to the miners as breaks, pinches, etc., which, as a

matter of fact, were usually barren where in contact with the older veins. (See Plate I.) But little encouragement was therefore offered by these veins, either to the surface prospector or to the miner underground. Many valuable mining claims of the district covering certain veins of this system remained idle and were considered as having but little value until they were cross-cut at deep levels from adjoining properties.

Structure of the Veins.—The Blue veins are typical fault fissures of marked displacement, with variable dips, but fairly uniform strike. (See Plates I. and II.) There is frequent splitting, or branching, on strike, the branches commonly re-uniting along the course of the vein after inclosing horses of granite. (See High Ore vein, Plate I.) In certain localities the branches are diverging and remain separate, as far as known from present developments. In closer detail the fault zone, which may be entirely included within a width of from 5 to 25 ft., is often composed of two or more well-defined movement planes characterized by seams of fault clay from 0.25 to 2 in. thick. The clay seams are usually accompanied by crushed granite of variable thickness, often occupying the entire width of the fault zone. The so-called crushed granite does not as a rule represent a cracked condition of granite adjacent to the movement planes, but more precisely, it is a distinct zone of finely comminuted granite, sharply separated from the fairly solid country rock by a wall or a clay seam marking a plane of movement.

Mineralogy of the Blue Veins.—The principal copper minerals are chalcocite, enargite, and bornite, together with small amounts of chalcopyrite, covellite, tetrahedrite, and tennantite, named in order of their importance. The gangue minerals are quartz and pyrite, with widely variable amounts of sphalerite and rhodochrosite, the latter minerals being present in certain localities only. Variations in the mineralogical composition of the ores of the Blue veins are common; in fact, are the rule. In certain ore shoots chalcocite predominates, in others enargite is the chief source of copper. Both are usually present, and bornite always but in less important quantities. Covellite is of small importance in the Gray Rock and Skyrme veins, but generally absent from the other veins of the Blue system. Chalcopyrite is of widespread occurrence, but in insignificant amounts, excepting in the Jessie vein, where it formed the chief ore mineral in the upper levels of the Jessie and Mountain Chief mines. Tetrahedrite and tennantite have been of little commercial importance, although of frequent occurrence, particularly in the northeast section. The gangue minerals are chiefly quartz and pyrite, with large

amounts of sphalerite in certain localities and minor amounts of galena, rhodochrosite, fluorite, etc.

Like the Anaconda veins, the mineral variations are matters pertaining largely to locality. As far as the primary copper minerals are concerned there is no possible mineralogical distinction between the ores of the Blue veins and those of the Anaconda system. The Blue veins lack the zone of sooty chalcocite, so common to the older veins. The minerals composing the ore bodies are believed to be almost universally of primary origin. Where shoots of primary ore reach the surface they are oxidized and secondarily enriched below the zone of oxidation in a manner similar to the Anaconda veins.

Ore Bodies.—The ore occurs in irregular shoots, either along the main fissure or along the various branches comprising the fault zone. On the strike of the fissure the ore shoots are separated by barren stretches of typical fault material consisting of crushed granite intensely altered and one or more clay seams. The altered granite usually carries abundant disseminated pyrite and frequently small wavy quartz-pyrite veinlets of small lateral extent.

The general distribution of the ore shoots is shown in plan on the accompanying map. (See Plate I.) It will be seen that they are irregularly spaced along the fissures, and, at this particular elevation at least, they appear to be arranged wholly without regard to Anaconda or other vein intersections. Typical examples of ore shoot occurrences are shown in Plates V. and VI., longitudinal projections of the Jessie and High Ore veins, respectively, upon which have been outlined the ore shoots (shaded areas) as far as known from present developments. The pitch of the ore shoots is usually to the southeast, but not uniformly so. The stope length of the shoots varies from 100 to 2,000 ft. The width of the ore runs from nothing up to 30 ft. or more. Stoping widths of from 10 to 20 ft. are common.

Structure of the Ore.—As a rule the vein filling of the Blue vein ore shoots is less massive than that of the Anaconda veins. A certain amount, however, of mineralization and replacement of firm wall rock has taken place along the Blue vein fissures, as well as in the older veins, resulting in hard, massive, complex aggregates of ore and gangue minerals similar to the typical " quartz-pyrite " ore of the Anaconda veins. Where, as is commonly the case, the ore is a complete or partial replacement of crushed granite, by quartz, pyrite, and copper minerals, it may be readily distinguished from ore formed by replacement of hard granite by reason of retaining the brecciated structure. Blue vein ore frequently consists of crushed and altered granite netted by stringers, bands, and tiny seams of the copper

minerals, chalcocite, enargite, bornite, and chalcopyrite. In such cases quartz and pyrite may be present in limited amounts, intimately associated with the copper minerals forming the ore seams. Crushed highly altered granite carrying disseminated chalcocite and bornite, are common features of Blue vein ore shoots.

The structural details of the ores are perhaps not as simple as might appear from the above general description. The order of deposition of the minerals has been much obscured by faulting, which took place along the fissures during the period of mineralization. The ore shoots often show rude banding due, in the main, to replacements along closely spaced fissures or planes of movement, or to shearing of the early-formed ore masses.

Influence of Vein or Fault Intersections upon Mineralogical Character of Blue Veins.—The intersection of a vein of the Anaconda system by a vein of the Blue system, or the faulting of a Blue vein by later fissures, has not resulted in marked changes in the mineralogical character of the Blue vein ore shoots. There is a possibility that at great depths the uprising solutions followed east-west fractures and at times found their way into the intersecting Blue fissures, through which they continued to ascend along the more open zones eventually forming the ore shoots. Developments are not of sufficient extent to determine this point conclusively.

The ore bodies of the Blue veins do not occur at intersections with older veins with any more frequency than in the wide stretches of barren country rock separating ths older veins. There is no evidence whatsoever that secondary ores have been formed in Blue veins through the action of descending meteoric waters in adjacent older vein segments, through which process the surface waters are supposed by Weed[10] to have taken up copper and, on their descent into the earth, spread out along the older vein and into intersecting Blue fissures, forming notable bodies of chalcocite. Plate VI., a longitudinal section of the High Ore vein typical of the Blue system, shows beyond question that the ore shoots bear no genetic relation to the positions of the intersected segment of the Anaconda vein.

Steward Vein System.

The mineralization of the Steward fissures has not been of such great economic importance as that of the older vein systems just described. The ore bodies of the No. 16 vein of the Rarus and Leonard mines are the largest yet found in this system of fissures. The La

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[10] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey,* p. 103 (1912.

Plata vein north of the Syndicate vein in the Mountain Con mine yielded a small amount of silver ore from shallow workings. In depth two small shoots in the Buffalo mine were worked for copper; however, they have thus far proved to be of little commercial importance. In the Gagnon mine the Gagnon South vein, apparently belonging to the Steward system, has yielded a considerable tonnage of copper-silver ore. This vein may possibly be the western extension of the No. 6 fault of the Parrot mine.

Occurrence of the Ore Bodies.—The ore of the Steward fault veins occurs in shoots similar in form and general character to those of the Blue vein system. There are many instances where Steward fissures are strike faults along veins of the Anaconda system, forming what might be properly termed compound veins. In examples of this class it is next to impossible to determine with any degree of certainty the influence of the fault fissure on the mineralogy of the older veins. Usually the fault fissure follows along the vein for a time on one wall, then crosses diagonally to the opposite wall. If the vein filling of the old vein is wide there may be but little breaking or crushing. As a general rule, the old vein itself is stronger than the adjoining altered wall rock, so that the strike fault occupies the plane of least resistance, which is usually the contact between the vein filling and wall rock. Further movement crushes the weaker altered granite, forming a zone of fault material following the solid ore. Where a small ore vein is overtaken by a strong strike fault both walls of the older vein may form planes of movement. The older vein filling thus inclosed within fault zones has the appearance of the usual fault vein ore and it is generally impossible to correctly classify such mineral masses without additional corroborative evidence. Even where the evidence proves conclusively that the major part of the vein filling belongs to a geologically older vein system, the influence of the strike fault on the mineral composition may be determined with difficulty, if at all. The mineral mass may exhibit later-filled fractures in primary vein filling, or replacements of the first-formed minerals, etc., but the phenomena of successive periods of mineral deposition are so frequently met with in veins of all ages that such evidence cannot be given great weight. The existence of minerals indigenous to the fault fissure itself in the vicinity of the mineral mass in question, identical with that found filling fractures in the older vein filling, offers satisfactory evidence of the fault influence.

The most important ore shoots found in fissures of Steward age and known positively to be indigenous are those of the No. 16 vein in the Rarus and Leonard mines. Although existing as a single well-defined

ore-bearing fault fissure in the Rarus mine, in the Leonard mine the No. 16 vein fissure splits into two or more branches, both of which contain important ore bodies. The ore is especially high grade, often showing from 1 to 4 ft. of reasonably clean chalcocite ore, containing also enargite, quartz, and pyrite, with little or no sphalerite, chalcopyrite, or bornite.

Depth of Ore Shoots.—No ore was found in the No. 16 vein higher than the 1,000-ft. level of the Rarus mine, or more than 1,100 ft. from the surface. In the Leonard mine the first ore was encountered at more than 1,200 ft. below the surface. These ore shoots have persisted to the greatest depths yet reached by mine workings.

Influence of Later Faults.—There is no evidence that the quality or mineralogical character of the Steward ores has been influenced by later faulting. The high-grade chalcocite-enargite shoot of the No. 16 vein in the Rarus mine is cut sharply by the Rarus fault unaccompanied by enrichment. (See Fig. 5.) Where the ores occur at great depth, as in No. 16 vein below the oxidized zone, it is doubtful if any appreciable amount of secondary enrichment has taken place. The chalcocite and other copper minerals of these deep ore bodies are believed to be universally of primary origin.

Minerals of Steward Vein Ores.—The mineral composition of the Steward ore bodies corresponds to that of the Anaconda and Blue veins, but depends on the particular locality in question. In the Rarus and Tramway mines the copper mineral comes principally from chalcocite, which occurs in great purity. There are usually present in small quantities enargite and bornite, but no sphalerite. There are also less quartz and pyrite than in the older veins. In the Gagnon district the ore shoots contain a large amount of sphalerite similar to older vein ores. The same is true of the La Plata ore shoots, where the ore contains sphalerite and rhodochrosite, both characteristic of the border zones.

In structural appearance the Steward ore shoots are similar to the previously described Blue vein shoots. They are, however, less complex both mineralogically and structurally, and it does not appear that the faulting movement continued to any great extent during the period of ore deposition.

CONTRASTING FEATURES OF VEIN SYSTEMS.

General Summary.

Forms of Ore Bodies.—The Anaconda fissures are solidly and continuously mineralized, forming ore bodies which may be continuously stoped over thousands of feet without showing any great variations

in valuable metal content. Anaconda veins seldom exhibit evidences of extensive fault movement so characteristic of the Blue and Steward veins.

The ore of the Blue and Steward fissures occurs in the form of "shoots" which vary greatly in size and extent. These ore shoots are irregular in outline and are separated on the strike of the fissure by hundreds or even thousands of feet of barren crushed granite and fault clay composing the fissure zone. The walls of the ore are seldom free from fault gouge or other evidences of extensive movement.

Oxidation and Enrichment.—The vein filling of the Anaconda fissures within the copper area extends with great strength upward to the oxidized zone. It is deeply oxidized and invariably shows a marked development of secondary chalcocite below the zone of oxidation, which has proved in most instances to be of immense commercial importance. (See Fig. 6A.)

In the Blue and Steward fissures the ore shoots seldom extend upward to within 500 ft. or more of the zone of oxidation. Except in the zones of highly altered granite the zone of oxidation in fault veins is shallow, the vertical extent being not more than from 20 to 40 ft., and often as low as 10 ft. The disseminated pyrite so universally present in the crushed granite of the fault fissure may show slight secondary chalcocite enrichment immediately below the zone of oxidation, but it is of no commercial importance. (See Fig. 6.)

Physical Character of the Ores.—As regards physical character, the ore of the Anaconda veins is usually harder, more massive, and it seldom exhibits the breccia structure so often characteristic of replacement of fault breccia of the fault veins. The solid ore streaks of the Anaconda veins are wider, and the metasomatic action has been sharper and more complete than in fault veins where vein-forming processes were often disturbed by fault movements.

Mineralogical Differences.—There are no notable differences in the mineralogy of the veins of the three systems in any given area. As noted on a previous page, the veins of a certain limited area contain minerals characteristic of that particular area, regardless of geological age. The Blue veins are later than the Anaconda veins and they have less pyrite but more quartz in proportion to the total vein filling present. There is also the same general order of sequence in mineral deposition in the various veins. The earliest minerals are quartz and pyrite, followed by enargite, bornite, and chalcocite in the central zone. In the intermediate zone the order is quartz and pyrite, enargite, sphalerite, bornite, and chalcocite, with chalcopyrite probably forming contemporaneously but under different conditions in

border zones. In the peripheral zone, the order among quartz, pyrite, galena, and sphalerite has not been worked out owing to inaccessibility of mine workings.

GENESIS OF THE ORE DEPOSITS.

Source of the Ores.

The evidence points directly to the conclusion that the ores have been derived primarily from rocks of igneous origin. The veins are found entirely within igneous rocks, with no sedimentary rocks in quantity within 15 miles of the district. Formerly sedimentaries may have covered the granite of the Boulder batholith, but there is no evidence indicating that such rocks played any part whatever in the formation of the Butte ores. The ore deposits of the district are so centered or concentrated within a relatively small area that causes which led to their formation must be sought within, or in close proximity to, the area in which the ores are found.

Compared with other mining districts situated within the borders of the Boulder granite batholith, the chief point of difference in the geologic structure is the presence in Butte of the quartz-porphyry dikes in close association with the veins. In the Corbin, Wickes, Rimini, Basin, Clancy, and Alhambra mining districts no quartz-porphyry has been found, although rhyolite intrusions are common, corresponding, no doubt, to the rhyolite intrusion forming Big Butte at Butte. The districts above named, however, have developed no important copper deposits, and it is believed, therefore, that the many rhyolite eruptions occurring at a comparatively recent date within the Boulder granite area, played a minor part in the deposition of the copper ores at Butte and elsewhere. The rhyolite at Butte is not directly associated with the ores in any way. The copper veins become poor going westerly, and the earlier vein systems are known to be older than the rhyolite.

There are three or more well-defined quartz-porphyry dikes traversing the Butte district in close association with the copper ores of the main central zone. Those dikes extend in an easterly and westerly direction, coinciding in a significant manner with the general habit of the veins of the Anaconda system. (See Plate I.) It will be observed that in going easterly from the Anaconda mine the Anaconda veins bend to the southeast toward the Silver Bow mine, while a similar bend may be noted in the quartz-porphyry dikes. Farther to the east a change in strike from southeast to northeast in the dikes is accompanied by a similar variation in the course of the Anaconda veins. These last changes in strike in both the dikes and the veins

are accompanied by corresponding changes in dips from south to north.

Another feature of interest in this connection is found in the fact that the areal extent of the quartz-porphyry appears to be increasing in depth. Where the upper levels of the Mountain View mine show but one small dike of an average width of 30 ft., the 2,200-ft. level discloses three dikes, having a combined width of 150 ft. The main St. Lawrence dike, found no farther west than the Nipper shaft on the surface, has been cut in the Gagnon, 1,900 ft. level, 2,500 ft. west of the Nipper. Again, in the Mountain View, West Colusa, and Leonard mines, areas of the most intense fissuring of Anaconda age are associated with quartz-porphyry dikes, and dikes wholly unsuspected at the surface are appearing in deep levels.

A certain significance might also be attached to the marked zonal arrangements of ore types about the central copper zone in which the more important quartz-porphyry dikes occur. This fact tends to support the belief that the Butte ores were derived from a demonstrable centralized source.

Notwithstanding these apparent close genetic relations between the copper veins of the central zone and the quartz-porphyry, it does not follow necessarily that the quartz-porphyry magma was the immediate and direct source of the ore minerals now found in the veins. The association of the quartz-porphyry dikes and the Anaconda veins seems to have greater significance when considered in connection with the origin of these early fissures than when referred to the source of the vein-forming waters. There is no evidence indicating a direct transfer of vein-forming waters from the quartz-porphyry dikes into the adjacent granite, nor does it appear that any notable alteration of the granite took place prior to the appearance of the Anaconda system of fractures. The dikes are altered only in areas where the granite is altered in connection with the veins. The Modoc dike extends northwesterly from the East Colusa mine through a long stretch of unaltered granite, in which the dike itself shows no alteration whatever. Other similar instances have been noted.

It is a reasonable assumption that the quartz-porphyry was derived primarily from the parent granite magma; and also that the ultimate source of the ore minerals was the granite magma. The principal part played by the quartz-porphyry has apparently been the opening of the way for vein-forming waters of deep-seated origin to reach the higher regions where the ore deposits are now found. While the ultimate source of the metals of the ores was probably the original

granite magma, the direct source may have been the same magma locally which furnished the quartz-porphyry, the latter rock following the earliest fracturing and at the same time stimulating an upward movement of the ore-bearing waters.

Ore Deposition.

Ore-Forming Agents.—There is little doubt that the chief transporting agent at work in the formation of the primary Butte ores was water. Vein-forming waters are believed to have been derived from the same fluid magma which earlier produced the quartz-porphyry dikes, and possibly also, at a much later period, the rhyolites. Such waters, of deep-seated origin, either contained primarily the elements which now go to make up the minerals forming the ores, or they gathered them from the wall rocks in the course of their upward journey. In regions open to observation the almost entire absence from the normal granite of many of the most abundant elements of the ore-forming minerals, such as copper, arsenic, sulphur, zinc, etc., points to the conclusion that these elements could not have been derived from the rocks adjacent to the avenues of travel, but that they were probably constituent parts of the solution when the upward journey began.

Composition of Vein-Forming Waters.—Certain general conclusions may be drawn as to the probable composition of these uprising waters when they reached the regions in which the ore deposits are now found, through a study of the character of the vein minerals known to be of primary origin, and the altered condition of the rocks directly associated with the veins. In the Butte district enormous quantities of copper, sulphur, arsenic, zinc, and manganese have been added, together with notable quantities of lead, antimony, silver, tungsten, barium, fluorine, etc.; most of which are very minor constituents of the normal granite rock and some of them have not been detected in the original granite by chemical analyses of fresh rock. In addition, the abundance of pyrite and quartz in the veins indicates the presence in the solutions of large quantities of iron and silica, both of which, however, are essential constituents of the original granite. Although purely a matter of speculation, it seems unnecessary to assume that the uprising solutions originally contained notable quantities of either iron or silica, both of which could have been readily derived from the granite wall rock at great depths through chemical processes.

The variable mineralogical nature of the ores of different veins presumes, apparently, many changes in the composition of the vein-forming waters from time to time. However, such a statement re-

quires some modification.   It is conceivable that from uprising waters carrying constant proportions of ore-forming elements, varying quantities of certain minerals might deposit at different localities within the same vein, or at different periods in the same locality, due to changes in temperature, pressure, or other controlling factors.   Analyses of the altered granite in association with the veins show that large amounts of sodium and calcium have been extracted from the granite.   These elements were undoubtedly present in the vein-forming waters in varying quantities along with the metallic elements which were later deposited as minerals in the vein.

The probable high temperature and vigorous chemical activity of the earliest solutions passing upward through the fissures of the Anaconda system is indicated by the intensely altered condition of the wall rock in the region of such fissuring.   The composition of these solutions can be judged in a general way only from a study of their effects.   The widespread development of pyrite both in veins and as disseminated pyrite in granite, in the absence of sulphur as a constituent of the original granite, requires the addition of sulphur in large quantities.   The extraction of the sodium and calcium in this process with addition of sulphur, forming pyrite, may be accounted for by the presence originally of an abundance of H_2S.   Since this activity was probably more intense at great depths, it is reasonable to assume that the ascending waters in the upper regions now open to observation carried not only hydrogen sulphide but alkaline sulphides as well.

Processes Involved.—The ore-forming minerals gathered at great depths, were transported upward and deposited largely through metasomatic replacement of the country rock along fissures so as to yield the ores now being mined.   The factors which influenced mineral deposition at certain horizons are believed to be mainly those of temperature and pressure.   However, another important factor entered into the vein-forming processes which is believed to account in no small way for the variable nature of the ore minerals in different localities and at different points in the same vein.   The association of the great areas of altered and pyritized granite with the massive quartz-pyrite veins belonging to the oldest vein system indicates that the early solutions were extremely active chemically, and it is not improbable that the first processes were in certain measure solfataric in their action.   These solutions or gases readily attacked the granite wall rock to which there was free access along the fissures, and they also found passage by means of cracks, joint planes, etc., outward for considerable distances from the main chan-

nels into the granite. While the alteration was in progress the oldest vein filling, largely quartz and pyrite, was being deposited along the fractures. In these early processes of granite alteration the attacking solution carried sulphur in some form. It is believed that at great depths it was, in part at least, in the form of hydrogen sulphide. Analyses of normal and altered pyritized granite show that, as a result of vein-forming action, the normal granite has lost sodium and calcium, but that in regions open to observation in the Butte mines, iron, although readily attacked, has remained fairly constant in quantity. The formation of pyrite in the process, accompanied by the extraction of sodium and calcium, would seem to indicate the presence of hydrogen sulphide. The action of the attacking vein-forming waters finally resulted in the formation of pyrite, largely *in situ*, from the iron of the original granite, and the soluble alkaline sulphide, Na_2S, which was not again deposited in the fissures, but migrated far beyond the zone of ore deposition, probably reaching the surface through hot springs. The presence of soluble alkaline sulphides in the vein-forming waters is believed to have exerted a marked influence in the deposition of the Butte ores, as will be later seen.

The above line of reasoning suggests the possibility that a large part of the pyrite, and possibly the quartz of the Butte veins, were not primary constituents of the vein-forming waters as they began their ascent from great depths. The presence of veinlets of pyrite in the altered granite in connection with the Anaconda veins indicates that some of the pyrite formed through the action of deep-seated waters on the iron of the granite, migrated for short distances at least, to be redeposited as vein pyrite. It is doubtful, however, whether appreciable amounts of the pyrite of the veins originated in this manner from nearby wall rock. Since, however, it is evident that the vein-forming solutions did attack the iron of the granite, it is not unreasonable to suppose that at greater depths conditions were such that the iron thus attacked did not immediately combine with sulphur to form pyrite *in situ*, but that the iron was taken into solution and remained so until more favorable conditions for deposition as pyrite were found. Such favorable conditions may have been encountered in the fissures at higher regions, now occupied by the ore deposits. The same method of reasoning may also be employed to account for the quartz of the Butte veins. The possible origin of the pyrite in the above manner has much significance when considered in connection with the variable mineralogical nature of the ores, as will be later set forth.

Further consideration of the possible chemical action between the uprising waters and the granite wall rock is of interest when viewed

in connection with the variations in mineralogical character of the ores deposited during different periods in the same locality, and also variations in ore types zonally arranged around the central copper zone previously described.

As already stated, it is believed that the earliest ascending waters vigorously attacked the wall rocks on their upward journey through the newly formed Anaconda fractures. Sodium and calcium were extracted from the normal granite, and since they were not again deposited, the attacking solution necessarily became more and more alkaline toward the surface. The chemical activity in the early stages was more intense because (1) the solutions were probably at a higher temperature than during later periods, and (2) when the fissures were first formed the solutions had direct access not only to the unaltered wall rocks of the fissures, but to much sheared and easily attackable, crushed, unaltered granite within and along the fissures. Under these conditions, if it be assumed that the original solutions contained certain proportions of hydrogen sulphide and alkaline sulphides, or alkaline carbonates, it is evident that the solutions would become more alkaline with a less preponderance of H_2S as they moved upward toward the surface.

From the state of maximum alkalinity, which is believed to have occurred when the early high-temperature waters had only unaltered granite to work upon, there was probably a gradual return to conditions wherein a higher ratio of hydrogen sulphide was established. Observed facts show that the alteration of the granite took place outward from the fissures, cracks, and joint planes through which the solutions passed. It is reasonable to infer that after a certain length of time the solutions no longer reacted chemically with the already altered granite adjacent to the avenues of travel; in other words, a partial chemical equilibrium was established between solution and the fringe of altered granite contiguous to the fissure, as far as the process concerned the extraction of alkalies or the attack of the iron of the granite. Metasomatic replacement of altered rock adjacent to the fissures by vein minerals, principally quartz and pyrite, was probably taking place during this time as a part of the general process. In the later stages, therefore, the ascending waters were unable to take into solution such large proportions of sodium, calcium, iron, and silica per unit volume of solution passing, owing to the presence of the protecting layer of already altered granite bordering the fissures, upon which the solutions had but a slight effect. Assuming then a fairly constant H_2S content of the original deep-seated solutions, there must have been at first an increase in the proportion of the alkalies and a

corresponding decrease in H_2S as the solutions moved upward, until a certain maximum degree of alkalinity was reached, after which time there was a gradual decrease in alkalinity, due to the protective effect exerted upon the normal granite by the altered granite barrier from which the available sodium and calcium had been extracted. A further decrease in the alkalinity of the solutions would be effected at later periods because of the development of solid veins, for subsequent fracturing of the vein filling (a common phenomenon) would permit a ready passage for the vein-forming waters along and in contact with vein minerals. The H_2S solution would thus be afforded a double protection against the influence of normal granite (1) by the vein filling, and (2) by the altered granite belt forming the vein wall rock.

If, as has been already suggested, the vein pyrite was derived from the granite through the action of uprising alkaline sulphide or H_2S solutions on the iron of the normal granite, the decrease in alkalinity would be accompanied by a corresponding decrease in the pyrite content of the later vein-forming waters, for the same reasons as above outlined. The latest vein filling in a given vein should contain less pyrite than the earliest concentrations of vein minerals.

Regarding the above discussion of the probable effect of the chemical action between the uprising vein-forming waters and the granite, as affecting the deposition of certain ore types, the writer does not assume that such reactions have been the sole influence in the determination of the vertical and lateral distribution of the minerals and ore types in the Butte deposits. In fact, this discussion has been undertaken largely for the purpose of bringing out more forcibly the possible genetic relations between certain ore types and the more important areas of altered granite. The elements Na, Ca, and Fe were chosen for discussion because comparative chemical analyses of normal and altered granite show clearly that these elements have entered prominently into the reactions between the vein-forming waters and the granite, and therefore, if such reactions were to any degree influential in the formation of ores, the above-named elements were, no doubt, the most directly concerned. To the variations in temperature and pressure, to changes in the composition of the original solutions from time to time, and to the mingling of unlike solutions must also be assigned important rôles in the distribution of the primary ores throughout the veins of the district.

Concerning the possible influence of these various factors as affecting the distribution of ore types, many facts have been observed that are of particular interest. For example, primary chalcocite and enargite occur only in association with the intensely altered granite of

the district. In the central copper zone, or in veins where alteration has reached an advanced stage, these two minerals are commonly found in intimate association with other vein minerals forming the ores, and, in addition, small veins or stringers, largely chalcocite and enargite, may be found extending outward into and replacing the wall rock, the granite being in actual contact with the copper minerals. In areas of unaltered granite, chalcocite and enargite are confined to highly altered zones within or along the veins, and never as stringers or disseminations inclosed by unaltered granite. It is improbable that solutions bearing these minerals did not find passage outward into the normal granite during the active vein-forming period, when chalcocite, bornite, and enargite were being deposited in abundance in the veins. The frequent occurrence of stringers of pyrite, quartz and sphalerite extending into wall rock of normal granite in the intermediate zone, with addition of manganese minerals in the peripheral zone, indicates that the mineral-bearing waters did to some extent find their way into the wall rock, but only where alteration was marked were chalcocite and enargite deposited contemporaneously in the veins proper and in the altered granite wall rock.

Applying a similar method of reasoning to the remaining important minerals of the veins, it is observed that quartz and pyrite are found abundantly in association with all stages of rock alteration and with all periods of primary ore deposition, although there is a noticeable decrease in the proportionate amount of pyrite in the intermediate and peripheral zones as compared to the central zone. It is true also that there is proportionately less pyrite in the Blue vein ores than in the Anaconda veins, and markedly less in the Steward ore shoots than in the Blue veins, an indication that the later vein-forming waters either contained less pyrite or that conditions were less favorable for deposition. These later ores are quite certainly less rich in pyrite for reasons previously stated. Quartz and pyrite are thus found to have been deposited under widely varying conditions of the solutions, such as composition, temperature, and pressure. Sphalerite is rarely found in zones of most intense alteration. It is sparingly developed in the slightly less altered areas of the Mountain Con and Diamond mines, and is very abundant in veins occurring in regions of relatively unaltered rock. It is believed to have been deposited under conditions of alkalinity similar to those under which enargite was formed, but generally at lower temperatures. Being more soluble, it would tend to migrate in solution before deposition through greater distances than would enargite.

Galena is a relatively uncommon mineral, associated usually with sphalerite. It is not found in the central copper zone. The manganese mineral, rhodochrosite, is an abundant constituent of the veins of the peripheral zone, and is not uncommon toward the outward limiting boundaries of the intermediate zone. It is entirely unknown in the central zone. Chalcopyrite is not an uncommon mineral associated with sphalerite, quartz, and galena in the manganese-silver veins.

In an endeavor to explain these general relationships between the various vein minerals and altered granite areas, certain assumptions must be made in order to form a working hypothesis. It is believed by the writer that the ores of Butte in their entirety have been derived from one general centralized source at great depths. There is no evidence in support of, and much evidence contradicting, the hypothesis that the so-called manganese-silver or zinc veins belong to a vein-forming epoch separate and distinct from that in which the copper veins were formed. Going outward from the central area (see Fig. 7) of typical zinc-free copper ores, there are no instances of early copper veins being cut by later veins of different mineralogical character, which would naturally be the case if, for example, the Blue fissures were of a distinctly later period than the copper veins of the east-west system. It has been suggested also that the manganese-silver ores represent an invasion of an older copper area, or *vice versa ;* this is also untenable, for the reason that it is an unlikely assumption that a later mineralization period could almost completely surround a central core, depositing new mineral in fissures identically the same age as those of the central area, without some of the later ore types appearing within the inclosed zone. It has been shown that the mineral composition of the different vein systems is not a result of separate periods of mineralization, but rather of geographic position. The structural relations point strongly to the conclusion that the east-west fractures of the manganese-silver area are of the same age as the Anaconda fractures of the copper area, and that they were well mineralized by quartz, pyrite, rhodochrosite, and sphalerite at the time they were cut by fissures of the Blue series. The Blue veins, furthermore, with their typical silver mineralization of the peripheral zone, continue southward into the copper area, cutting the oldest copper veins, and there contain valuable copper ores and are characteristic copper veins. Fissures of the Steward system are also found in both the copper and the silver areas, cutting copper ores in the former case and zinc ores in the latter. Therefore, while some copper ore bodies are found in these fissures, and zinc mineral-

ization is also found in them, the bulk of the vein filling in both areas is prior to this system of fissures. These structural relations between mineral deposition and the faulting periods point to the conclusion that the primary ores of the whole district are products of one great vein-forming period beginning immediately after the appearance of the Anaconda fractures, and ending with the completion of the ore bodies of the Steward vein system. It is not improbable that during this period there were some interruptions and possibly at times ore deposition practically ceased, or that vein-forming action was more vigorous in some portions of the district than in others. The fracturing of early-formed ore with subsequent filling of such fractures with new minerals is not believed to mark distinct periods of vein-forming action, but rather to indicate faulting movement during the active deposition processes. It is reasonable also to infer that the older fissures became partly or wholly plugged or sealed by minerals, at certain places. The result of subsequent fracturing by faulting might merely divert ore-bearing waters from localities where they were yet active into new fractures in the formerly plugged portion of the veins.

Proceeding on the assumption that the primary vein minerals were derived from a common source at relatively great depths below where they are now found, it is believed that the broad, general, orderly arrangement of the ore types both vertically and laterally is due in a large measure to the variable temperature and pressure conditions encountered by the vein-forming waters along the lines of travel toward the surface, and to changes effected in the chemical composition of these transporting waters through their action on the granite wall rock. Moreover, there were, no doubt, wide variations in the metal content of the vein-forming waters from time to time. It is believed that the minerals of the veins were more soluble in strongly alkaline sulphide solutions than in hydrogen sulphide solutions under conditions of constant temperature and pressure, but in either alkaline sulphide or hydrogen sulphide solutions they were more soluble in hot than in cold waters.

The areal distribution of the mineral types found in the oldest vein system (the Anaconda) is believed to be due, in part, to the relative solubility of these various minerals in the original uprising waters from which they were deposited. It has been previously pointed out that these early solutions were carriers of alkaline sulphides due in part to the extraction of sodium and calcium from the granite. Under conditions of high alkalinity and elevated temperature, quartz and pyrite were deposited in abundance, forming the massive quartz-

pyrite veins of the central and intermediate zones. Under high temperature conditions, the minerals, sphalerite, rhodochrosite, galena, and chalcopyrite were more readily soluble and migrated outward to the intermediate and peripheral zones, where they were deposited along with quartz and pyrite, forming the primary vein filling of the so-called manganese-silver veins. During the period of migration of these solutions from the hotter zones the solutions lost none of their alkalinity, but the temperature was greatly reduced. These conclusions are drawn from the fact that there is no evidence to indicate that calcium and sodium were precipitated; in fact, slight alteration of the normal granite along the veins indicates a further addition of these elements to the vein-forming waters; and furthermore, the probable occurrence of alkaline carbonates is shown by the abundance of rhodochrosite in the border zones. The lower temperature conditions and comparative inactivity of the solutions toward the granite in the outer regions are well shown by the slight alteration of the wall rock adjacent to the fissures. In great veins a hundred feet or more in width in the manganese-silver district, alteration of the granite extends for only a few feet outward from the general vein boundaries.

These early solutions contained some copper, as shown by its almost universal presence in the oldest known quartz-pyrite vein material and in the disseminated pyrite of the altered granite. Apparently no copper minerals were deposited in quantity in the earliest stages of vein formation, except possibly the chalcopyrite of the manganese-silver veins, but enargite is known to have formed in considerable abundance in the Anaconda veins prior to the appearance of the Blue fracture system, not, however, until the old veins were well formed. Enargite is therefore regarded as a relatively high-temperature mineral, but it probably formed under less alkaline conditions than existed in the early stages of the process, when quartz and pyrite were first deposited. The absence of notable quantities of other minerals containing copper or arsenic associated with this early enargite vein filling implies that these two elements were present approximately in the necessary proportions to form enargite. Copper was probably in excess, otherwise arsenopyrite would have been formed in cooler regions. Chalcocite did not form from the copper excess owing to the alkalinity of the solutions. It is probable that some bornite was formed at this early period, as was chalcopyrite in the outer zones of lower temperatures.

During the later stages when the Blue and Steward fault vein ores were formed, enargite, bornite, and chalcocite were deposited in large quantities in veins of all ages within the central and intermedi-

ate zones. In all of these veins the same order of deposition is noted; that is, pyrite and quartz are the oldest minerals, followed by enargite, sphalerite, bornite, and chalcocite, in the order named, chalcocite being associated with the older vein minerals only where the granite is intensely altered.

The general relationships above noted between the various ore types and conditions of granite alteration tend to show that under conditions of high alkalinity and high temperature and pressure, the vein minerals first deposited were principally pyrite and quartz. That this condition prevailed in the early stages of mineralization in fractures of all ages, except where the later fractures passed through areas of granite previously altered, is shown by the universal priority of a portion of the quartz and pyrite. As previously stated, the solutions became less alkaline, with a corresponding increase in the proportion of hydrogen sulphide present. It is known that the metals common to Butte, such as iron, copper, zinc, and manganese, are soluble in sodic sulphide solutions, but less so in hydrogen sulphide solutions; in fact, when present in sufficient proportions, hydrogen sulphide acts as a precipitant for these metals. The change of solution from a highly alkaline condition to one in which hydrogen sulphide predominates is brought about through conditions approaching chemical equilibrium between the uprising solutions and the granite wall rock. That chalcocite is a late mineral in the ores is well known. The explanation may be found in the suggestion here offered that it was deposited in the Butte veins only when the amount of hydrogen sulphide present in the vein-forming waters was large in proportion to the alkaline sulphides present. Observed conditions in the veins tend to support this view. Regardless of the geologic age of the fissure, quartz and pyrite are found to be the first minerals to form, chalcocite was one of the latest, and enargite, sphalerite, and bornite were found in intermediate stages. Quartz and pyrite, however, are formed under all conditions and, therefore, are intimately associated with minerals of the later stages, including chalcocite.

The deposition of chalcocite, the cuprous sulphide of copper (Cu_2S), directly from ascending waters, seems well within the range of possibility, considering that it is well known, or at least it has been frequently stated,[11] that the minerals enargite[12] and bornite, admittedly primary[13], contain the cuprous sulphide molecule Cu_2S.

[11] Dana's *System of Mineralogy*, pp. 76 and 147 (6th ed., 1892).

[12] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, p. 82 (Jan., 1912).

[13] Graton, L. C., The Sulphide Ores of Copper, *Trans.*, xlv., 26 (1913).

With an excess of copper present in the cuprous form, over and above that required to satisfy the enargite molecule, it would seem reasonable to look for the deposition of chalcocite or bornite under favorable conditions.

Many minerals have undergone alteration, particularly the copper minerals, enargite and chalcocite. These alterations have been effected by primary processes. Enargite frequently alters to or is replaced by chalcopyrite and bornite, and chalcocite commonly exhibits slight alterations to bornite and rarely to chalcopyrite. Such alterations are believed to be due to the instability of the minerals under changing conditions in the solutions.

Summary of Ore Genesis.

The original source of the ores at Butte was the granite magma. Quartz-porphyry dikes formed a local closing phase of the igneous activity connected with the intrusion of the parent rock, and these dikes structurally and areally are in such close association with the ore deposits that they appear to be a direct factor in the localization of the ores. Heated waters and gases escaping from the cooling magma were the carriers of the metals to their place of deposition. The elements thus transported and deposited in the veins were silicon and oxygen as SiO_2, sulphur, iron, copper, zinc, manganese, arsenic, lead, calcium, tungsten, antimony, silver, gold, tellurium, bismuth, and potassium. Small quantities of potassium are believed to be added to the granite in the sericitization process.[14] Other elements, as sodium, calcium, and manganese, were undoubtedly carried by these solutions, but, as shown by analyses, they were extracted from the granite in the alteration process instead of being added as in the case of the first-named elements.

The chemical composition of these ascending waters varied in significant particulars as the process progressed. The granite wall rock was decomposed, furnishing much sodium, calcium, and possibly magnesium to the solution. Iron was also freed from the iron minerals of the granite to form pyrite with the sulphur of the invading waters. These interchanges affected the solvent capacity and character of the ore-bearing waters by the subtraction of the acid radical sulphur and the addition of alkaline radicals. While hydrogen sulphide and acidic conditions may have prevailed at the initial stages of ascent, the waters would tend to become alkaline through interaction with the wall rock. Along circulation channels, however, this

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[14] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, p. 67 (Jan., 1912).

action would gradually become less pronounced after a barrier built of sericitized granite had been formed bordering the fissures, thus protecting the solutions from further reaction with the fresh granite, and permitting the acidic conditions to ascend to higher horizons. Also, the earliest vein minerals, chiefly quartz and pyrite, would tend to insulate the solution from the granite. And finally, increasing alkalinity of the solutions and lower temperature would lessen action on the granite at points further removed from the central source.

Applying the above reasoning to the facts of ore occurrence, it is found that chalcocite as a primary mineral is the latest important copper sulphide of the ores; it is, moreover, found only in association with the highly altered phases of the granite. From these facts the conclusion may be drawn that under the geologic conditions existing in Butte, the more acidic conditions were necessary for the deposition of this mineral. Similarly, enargite is associated with highly sericitized granite, and is therefore believed to have been deposited only under certain conditions pertaining to the temperature and relative alkalinity of the solution.

Sphalerite, rhodochrosite, and galena are increasingly abundant toward the intermediate and peripheral zones, suggesting their formation under lower temperature conditions with relative high alkalinity. Quartz and pyrite are everywhere present, and evidently are formed under all conditions. Pyrite is more abundant in the central and intermediate zones than in the peripheral zone. Quartz is more prominent as a gangue mineral in the peripheral zone than elsewhere.

Structurally there is no good evidence for distinct periods of mineralization in the Butte veins. It is here held that there was but one period of mineralization, varying in intensity, possibly, from time to time, with important changes in chemical character of solutions. But the mineralogical difference in vein material of the central, intermediate, and peripheral zones can be adequately explained, it is believed, by the reasoning herein set forth, which assumes that the copper mineralization indicates high temperature and acidic conditions versus lower temperature and alkaline conditions as the solutions migrated toward the peripheral fractures now represented by the manganese-silver veins.

Concerning the formation of chalcocite there is much geologic evidence, mainly structural, to support the theory above outlined, which assigns to this mineral a primary origin from deep-seated waters. The subject of chalcocite formation is of exceptional interest and well deserves special treatment in connection with the geology of the Butte

copper deposits. The evidence which tends to support the primary chalcocite theory held by the writer is briefly outlined in the chapter which follows.

Origin of the Butte Chalcocite.

Owing to the persistence to great depths of the mineral chalcocite in the Butte copper veins much interest has been aroused among geologists concerning the manner in which it was formed. In recent years the opinion has been quite generally held that chalcocite is largely, if not wholly, a product of descending sulphide enrichment. This view arose naturally through the discovery of the so-called " black sulphurets " (later proved to be sooty chalcocite) of Ducktown, Bisbee, and similar pyritic ore bodies. These belts of black amorphous chalcocite were found separating the oxidized zone from the lean pyritic ore below and they were early believed to have resulted from the reaction between the descending copper sulphate waters and the unchanged primary ores below. That this view was the correct one for the sooty chalcocite of this class of deposits has been abundantly proved by recent investigations.

The discovery of similar chalcocite ores in the early mining operations at Butte led many observers to the opinion that these remarkably rich ores were likewise of secondary origin and of limited vertical extent. When the zone of sooty chalcocite was penetrated, however, the predicted lean cupriferous pyrite ore was not found, but chalcocite-bornite-enargite ores were encountered, which have persisted to great depths. The chalcocite of the deeper levels does not occur in the sooty form, but instead, it is the gray massive mineral more or less intimately mixed or intergrown with bornite, enargite or other ore minerals replacing directly altered granite. It is not necessarily a replacement of pyrite or any other sulphide mineral, being deposited directly from solution as chalcocite in veins along with bornite and other copper sulphides.

The problem of the formation of the chalcocite in the Butte veins was studied recently by C. T. Kirk,[15] who endeavored to work out a definite relation between the chalcocite deposition and certain stages of granite alteration. He concludes that such a relation exists, and that the chalcocite formation is, in the main, associated with a certain phase of granite alteration which has developed through the action of descending meteoric waters. Weed,[16] in his recent report on Butte,

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[15] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (January, 1912).

[16] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S Geological Survey*, p. 76 (1912).

likewise declares that most of the chalcocite has resulted from descending waters, although primary chalcocite also occurs, but just how and where he fails to state.   Many other writers familiar with these ore deposits, notably H. V. Winchell and the late S. F. Emmons, regarded the chalcocite as chiefly of secondary origin, at the same time holding the view that some of it might be primary.   Recently, however, Winchell [17] has expressed the view that the deep chalcocite is largely primary.

An intimate acquaintance with these ore deposits extending over a period of years has led the writer to the conclusion that most of the massive chalcocite is of primary origin, in the sense that it was deposited in its present position directly from deep-seated ascending solutions.   Secondary chalcocite exists in large quantities also, but it is believed to be of limited vertical extent, being confined principally to the well-known sooty chalcocite zone extending from the bottom of the oxidized zone to depths ranging from 100 to 1,200 ft.   It should be clearly understood, as previously stated, that the sooty glance zone has no well-marked lower limit, and futhermore, in the generation of sooty glance by descending waters, massive chalcocite is frequently developed, especially where the replacement of pyrite or other sulphide has reached an advanced stage.   It is impossible to differentiate in hand specimens between primary and secondary chalcocite when both appear in massive form.   There is of necessity in many cases an overlapping of primary and secondary chalcocite in the veins, inasmuch as primary chalcocite is believed to have originally extended to an elevation higher than the present ground surface.   It follows, therefore, that to some extent primary chalcocite has been subjected to the action of atmospheric agencies along with the associated primary vein minerals.   The result has been a sooty glance enrichment of the primary minerals of the ore, among which there existed massive chalcocite.

Formation of Primary Chalcocite.—The observed facts which have led the writer to the conclusion that primary chalcocite exists in large quantities in the Butte veins may be briefly stated as follows:

1. The occurrence of chalcocite in great abundance at levels 3,000 ft. or more from the surface.

2. The intimate association of chalcocite with bornite, pyrite, and enargite in such a manner that all must be regarded as having been deposited at the same time and under similar conditions.

3. Chalcocite is found at all depths without regard to surface to-

[17] Winchell, H. V., Discussion of L. C. Graton's Paper, The Sulphide Ores of Copper, *Trans.,* xlv., 84 (1913).

pography, which fact tends to show that no relation exists between the occurrence of chalcocite and present-day downward-seeping waters.

4. Chalcocite occurs in absolutely dry veins and ore-shoots at deep levels, and in many instances large bodies are cut by older faults, a fact further tending to show that this copper mineral is an old one and in no way genetically related to the present-day or a former similar underground water circulation.

5. Chalcocite directly replaces altered granite at deep levels. The power of cold meteoric waters to effect direct replacement of granite in quantity is seriously questioned by the writer.

6. No evidence is available tending to show that chalcocite is now being deposited in the veins, except within the sooty chalcocite zone. On the other hand, where positive evidence on this point is obtainable, it indicates a tendency of the massive chalcocite to alter to bornite and chalcopyrite under present ground-water conditions.

As outlined in the discussion of the formation of sooty chalcocite, the facts plainly show that secondary chalcocite has resulted from downward-seeping sulphate waters, and there can be no doubt that this mineral was in the active process of formation at the time the first mine openings were made in the copper veins. Concerning massive chalcocite of the deeper levels, however, there are important reasons for believing that it is in no way related genetically to the present existing meteoric ground-water circulation, or with any water circulation system of meteoric origin, but that, whatever the source, its time of formation must be referred to a relatively old mineralization period. Reference to Plate II. will assist in making this point clear. It will be seen that the Rarus fault sharply cuts all the important ore veins, displacing them hundreds of feet, so that in the intersected veins the possibility of surface waters effecting an enrichment of the truncated portion of the veins lying beneath the fault is extremely remote. The upper displaced segment of the O'Neill vein, for example, is no richer in chalcocite than the sub-fault segment, excepting within the chalcocitization zone directly beneath the oxidized zone. It is evident that the descending sulphate waters moving down the upper segment could not possibly reach the lower segment, and there is no indication of an enrichment of the upper segment where it meets the Rarus fault. The moving waters did not enrich the fault, as it carries no ore, neither did these solutions spread out to other veins cut by the fault. What is said here regarding the O'Neill vein is equally true of all the veins intersected by the Rarus fault. In examples of this character where the possible source of the supply of

the sulphate waters has been effectually cut off from the lower parts of the veins by intervening faults, it becomes evident that the chalcocite of the lower segments either had its source in uprising solutions or else it was deposited from a descending water circulation existing long prior to the appearance of the fault, and possibly far removed from the conditions as we now know them.

Following this line of reasoning, it is possible, as will be later shown, to prove that chalcocite, other than the sooty variety, is a comparatively old mineral, and that it was deposited in great quantities prior even to the faults of the Steward system, which in themselves carry important bodies of ore composed of enargite, bornite, sphalerite, barite, galena, and other well-known primary minerals.

In the No. 16 vein, a mineralized fault of the Steward system of the Rarus and Tramway mines, extensive bodies of chalcocite-enargite ore are sharply cut by the Rarus fault. (See Fig. 5.) These ore bodies are in the form of the characteristic fault-vein ore shoots and actual development proves that they do not extend upward to within 1,100 ft. of the surface, the higher portions of the fissure being absolutely barren of ore or gangue minerals. Like the chalcocite ores of the O'Neill vein above noted, the No. 16 vein ore shoots were formed long prior to the Rarus fault. The altered condition of the crushed zone of the Rarus fault, the presence of much disseminated pyrite and quartz, together with the fact that a later fault (Middle) cuts and displaces the Rarus fissures, tend to show that geologically the Rarus is not a recent fissure, therefore the water circulation responsible for the chalcocite older than the Rarus fault must be far removed from the meteoric ground-waters of to-day.

Going further into the history of chalcocite, certain facts seem to indicate, if not definitely prove, that chalcocite existed as a vein mineral prior to the Steward fault period. In the ore breccia of the Gagnon mine fragments of older vein matter containing chalcocite are of common occurrence. The Steward fault fissure is of later origin than the breccia and the breccia is much squeezed and faulted where they come in contact. These angular ore fragments are within and form a part of the original breccia and they are plainly not of secondary origin. They are not breccias resulting from Steward faulting or any other fault movements, but they were formed in the same manner as the Mountain View breccias and probably at the same time. These ore fragments represent a period of mineralization of an earlier date, and they are not drag ore, but pieces of older vein which have fallen into open cracks.

In tracing the formation of earlier chalcocite, attention must be

given to the remarkable ore shoots of the Blue vein fissures, of which the great ore bodies of the Jessie, Edith May, High Ore, Skyrme, and Blue veins are examples. It is a significant fact that many of the largest and most important rich chalcocite-enargite ore shoots in these veins do not extend upward to within from 500 to 800 ft. of the surface. (See Plate VI.) Not only do the copper minerals fail, but the common gangue minerals, quartz and pyrite, drop out, so that the ore shoots are capped by hundreds of feet of barren crushed granite and fault clay marking the plane of movement. In two instances, notably in the Jessie and Blue veins, ore shoots reach the surface, but in these cases the upper 500 ft. of the shoots differ materially in mineralogical composition from the richer ores at greater depths.

A study of the composition and structure of these remarkable ore shoots indicates that the minerals forming them have had a common origin. They are not connected or related in any manner with cross-fissuring or later faulting. A glance at Plate I. will be convincing on this point, that the Blue vein ore shoots do not occur at the intersections with older quartz-pyrite veins, but on the contrary, curiously enough, they are almost universally found in the intervals between the important older veins. A marked uniformity in alignment may be noted in the shoots in a northeast-southwest direction. No apparent relation exists between the Blue vein shoots and the later Steward faults; in fact, repeated observations of such intersections show beyond question that the ore shoots were in existence prior to the appearance of the Steward faults. It is next to impossible, however, if not entirely so, to determine what amounts, if any, of the minerals composing the Blue vein ore shoots were added at a period immediately following the appearance of the Steward fissures. Ore-bearing solutions traversing the Blue fissures, after the Steward faulting began, did not necessarily originate through the later fractures, nor did solutions passing along Steward fractures necessarily find their way into the Blue veins. In both cases the circulation was confined largely to irregular zones within the fissures themselves, which are now marked by the positions of the ore shoots. The intersections of Blue vein ore shoots rich in chalcocite by Steward fissures are numerous, and, from the evident lack of influence on the mineralogical character of the ore, the writer is led directly to the conclusion that chalcocite did exist in large quantites in the Blue vein prior to the Steward faults.

Assuming for the moment that the above inference is the correct one, the difficulties met with in an attempt to ascribe a secondary origin to this early chalcocite are numerous and of vital import. At the close of the Blue vein period (which period is assumed to be the

time elapsing between the beginning of Blue vein movements and the beginning of Steward faulting), it is fair to assume that the ground surface was much higher than at present, necessitating, therefore, a former extremely deep meteoric ground-water circulation to reach chalcocite ore bodies of the Blue veins now found more than 3,000 ft. from the surface. When one considers the rate of the downward invasion of the oxidized zone, it is almost inconceivable that down-seeping sulphate waters could have formed the extensive chalcocite ore bodies found at these depths. The time required would be enormous, and, furthermore, the fact must not be lost sight of that under conditions favorable for sooty chalcocite formations, as we know them, a very large part indeed, if not all, of the copper of the descending sulphate waters is deposited as secondary chalcocite before a maximum depth of 1,200 ft. below the zone of oxidation is reached.

There is another important point inviting attention, relative to the probable condition of the underground circulation existing during the time of formation of the Blue vein ores and during subsequent periods extending to the present time. It is a self-evident fact that meteoric waters could not have descended to great depths along veins, faults, or fissures at a time when appreciable quantities of waters, presumably deep-seated, were ascending through such channels. It is a reasonable assumption, then, that no important downward movement of meteoric waters took place in the Butte fissures until after the cessation of movement of the uprising solutions from which were deposited the primary ores. It is not unreasonable to believe that some surface waters did reach these channels of uprising waters at comparatively shallow depths, not, however, by direct descent along fissures through which deep-seated waters were ascending, but by a downward-lateral movement through neighboring fissures adjacent to the main trunk channels. As has been formerly pointed out, however, the movement of cold surface waters through normal granite is scarcely appreciable, and it is therefore extremely improbable that such waters could have influenced chemically, physically or in any way whatsoever the action of the deep-seated waters as they moved upward through the fissures depositing minerals undoubtedly derived from deep-seated sources. The occurrence of undoubted primary ores, or quartz, pyrite, sphalerite, galena, and rhodocrosite, together with enargite, bornite, and chalcocite, in faults of the Steward system which are known to cut and displace chalcocite ore bodies in the Blue and older vein systems, is conclusive proof that ascending solutions depositing primary ore continued in action long after the formation of the Blue vein chalcocite. It is probable that ascending waters

continued to traverse the Steward and older fissures for a considerable length of time after the primary Steward ores had formed; in fact, it is not at all improbable that the alteration of the granite and deposition of pyrite in the Rarus fault resulted from ascending waters.

If an early circulatory system existed similar to that above assumed, it is difficult to understand how meteoric waters could have been active enough to transport large quantities of mineral from the oxidized zone to great depths at any period prior to the complete cessation of the upward movement of the primary ore solution. The time of cessation must have been as late as the end of the ore-forming period of the Steward fault veins, and possibly as late as the Rarus fault, both of which are known to be later than much of the chalcocite of the Blue and Anaconda veins.

The chalcocite of the deeper levels, or in a general way the massive chalcocite of all the veins, bears no definite relation to the present surface topography. This is in marked contrast to the occurrence of sooty chalcocite known to be of secondary origin. The tops or apices of many rich, massive, chalcocite ore bodies or shoots are found at depths ranging from 100 to 1,500 ft. from the surface. The size and richness of the ore body are in no way indicated by the depth of the zone of oxidation; in fact, many of the largest ore shoots of the fault veins are capped by from 500 to 800 ft. of barren crushed granite and fault clay having an oxidized zone of less than 25 ft. in vertical extent. In the quartz-pyrite veins of the Anaconda system where the development of secondary glance is greatest, there is an apparent close relation between the depth of oxidation and the quantity of sooty chalcocite found below. A deep zone of secondary chalcocite is certain to be found below a deep zone of oxidation, while a shallow zone of oxidation is accompanied by an unimportant development of sooty chalcocite below. If a secondary origin is assumed for the chalcocite ore bodies whose tops or apices are separated from an extremely shallow oxidized zone by hundreds of feet of barren crushed granite, the question as to the source of the copper to form the chalcocite becomes of vital interest.

Where the ore shoots do not extend upward to the oxidized zone it does not appear possible that the source of the chalcocite could have been at a point higher than the top of the ore shoot, for there is no evidence, direct or otherwise, that copper-bearing mineral of any character ever existed in the eroded and oxidized portions of the vein. The marked absence, in the upper portions of many veins, of an adequate source of supply for the copper found at great depths in the form of chalcocite is a common feature of these ore deposits. This

is true not only of the fault veins, but in many of the veins of the Anaconda system. In the Tramway and Leonard mines, for example, immense chalcocite-enargite ore bodies from 50 to 200 ft. in thickness, belonging to the Anaconda vein system, have been developed between the 1,200 ft. and 2,000-ft. levels. From the 1,200-ft. level to the surface these ore bodies are represented only by small veins from 2 to 6 ft. in thickness, carrying but small amounts of copper. Indeed, in many instances the identity of the vein is entirely lost as the higher levels are approached. In nearly every case the oxidized zone capping the big ore bodies of this section is shallow, and even if it be assumed that hundreds of feet of vein have been eroded, such eroded portions represent a source entirely inadequate to account for the chalcocite found below.

In the Shannon vein (belonging to the Anaconda system) of the West Colusa mine, the ore bodies of the upper levels were of tremendous size, particularly in the region immediately underlying the oxidized zone, where there occurred a big development of sooty chalcocite. It is significant that although the vein continued big and strong in depth, with every condition favorable for secondary enrichment, the vein became poor rapidly in depth, portions of it at the 900-ft. level being too low grade to mine. Many other examples might be mentioned where old quartz-pyrite veins have been broken by later faults, and all conditions seem ideal for the formation of chalcocite ores in depth, but, other than the sooty glance enrichment, no notable addition of chalcocite has taken place.

An interesting occurrence of chalcocite is found in the Mountain Chief ore shoot of the Jessie vein, belonging to the Blue fault system. Plate V., a longitudinal projection of the vein, has been prepared to show the forms and positions of the various ore shoots. As will be noted, the oxidized zone is extremely shallow, being not more than 25 ft. deep at any point in the vein. The ore shoots have been opened by continuous workings from the surface to the 2,200-ft. level.

There is a marked difference in mineralogical composition of these ore shoots between their upper and lower portions. The change is found to take place at a depth of from 500 to 800 ft. from the surface. Some of the shoots do not extend entirely to the oxidized zones. The line $A-B$ is drawn to mark approximately the elevation at which the change takes place. Above $A-B$ the ore is an intimate mixture of quartz, pyrite, and chalcopyrite, the latter mineral occurring in abundance. Sphalerite and rhodochrosite are also present in considerable amounts. In the Mountain Chief mine one shoot extends entirely to the surface, where it is oxidized to a rich ore com-

posed of cuprite and iron oxides. Immediately below these rich oxides there is but a slight development of secondary chalcocite, occurring as thin films coating the chalcopyrite and pyrite. At about the line $A-B$, within a vertical distance of from 50 to 75 ft., there is almost a complete transitional change from chalcopyrite ore to an ore consisting of chalcocite, bornite, enargite, pyrite, and quartz, with only small amounts of chalcopyrite. This character of mineralization has continued to the deepest levels yet opened, although there are some variations in the relative amounts of the minerals present. The development of chalcopyrite seems to take place only in the high levels or at the waning ends of the ore shoots, indicating possibly that under certain conditions it is a lower-temperature mineral than either enargite or chalcocite, assuming for the moment that all three are here of primary origin.

In this particular example it is impossible to conceive of a surface water origin for the chalcocite lying below the chalcopyrite capping. There certainly is no apparent adequate source for the copper. Where the chalcopyrite ore suffers oxidation the larger part of the copper is held in the oxidized zone as an oxide or carbonate, and, even assuming that a part of the copper was carried downward, it is quite impossible for the writer to believe that it could have remained in solution while passing downward over the chalcopyrite-pyrite ore, to be later deposited as chalcocite from 800 to 2,200 ft. below the surface.

As already stated, many of the Butte veins have but slight oxidized zones, accompanied by sooty chalcocite zones of small vertical extent separated by hundreds of feet of barren vein from the chalcocite ores below. In such instances it is impossible to trace any genetic relation between the meteoric water circulation and the chalcocite commonly occurring at depths greater than 1,000 ft. Where, however, as in the case of most of the quartz-pyrite veins of the Anaconda system, an important chalcocitization zone exists associated with massive chalcocite ores, and is underlain at greater depths by large quantities of chalcocite not associated with sooty chalcocite, it is, perhaps, reasonable upon first thought to suppose that all the chalcocite of both higher and lower levels has had a common origin. The early prominence of the copper veins of this class has been largely responsible for the former general belief in a secondary origin for the chalcocite in the Butte veins.

The changes which occur in the oxidized zone of the old copper-bearing quartz-pyrite veins in Butte are due to processes which act slowly. The invasion of the oxidized zone downward into the sulphides took place at an extremely slow rate, and in view of this fact it

must be admitted that the sulphate solutions originating at the sharp contact between the oxides and sulphides were extremely dilute. The chemicals added to these downward-seeping surface waters through the oxidation of the sulphides are chiefly copper and iron sulphates, and possibly small amounts of free sulphuric acid. These cold surface waters, after taking up their burden at the oxide-sulphide contact, pass immediately downward along the vein, or partly through country rock, moving more or less constantly in direct contact with the pyrite and other vein sulphides. As is well known, the reaction between the sulphides and sulphate waters results in the formation of sooty chalcocite as a direct replacement of the sulphide attacked. These sulphate waters also attack the altered granite, resulting in greater porosity and in the formation of abundant kaolin. There is also a chalcocitization of the disseminated pyrite so common in the altered granite. In meeting already existing ground-waters below, a large proportion of which, although of meteoric origin, did not take copper into solution on their downward journey, the descending waters along the veins must become more and more dilute and certainly less active chemically as greater depths are reached. As a matter of fact, actual comparisons of the veins and granite of the upper and lower mine levels show conclusively that the downward-seeping waters actually become weak and inactive at not great depths below the surface, and it was due to this fact, in part at least, that C. T. Kirk was able to differentiate so clearly between the chloritic, sericitic, and kaolinitic alteration phases in the Butte granite.

It is extremely important to understand clearly this feature, because, apparently much more vigorous chemical processes have been active in the formation of the massive chalcocite of the deeper levels than were necessary for the formation of the secondary chalcocite of the higher levels. In the sooty chalcocite zone only sulphides are attacked and replaced by the chalcocite, while at greater depths massive chalcocite alone, or intimate mixtures of chalcocite, pyrite, bornite, and enargite, directly replace altered granite in quantities within and along the fault veins and veins of the oldest system. The writer believes that such replacements could not have been effected by dilute meteoric waters, which, in the act of reaching great depths, not only became extremely dilute and of doubtful activity, but they have been deprived wholly or in part of their copper in the regions of sooty chalcocite formation.

The observed facts which have led primarily to the belief that the chalcocite of the Butte copper deposit is of secondary origin may be briefly stated as follows:

1. The chalcocite is often of a later age than the vein minerals with which it is associated.

2. In some instances the proportionate amounts of chalcocite in the veins have decreased rapidly with depth.

3. It has been abundantly proved at Ducktown, Morenci, Bingham, and in many other instances, that under certain conditions chalcocite is a product of descending sulphate waters.

4. The depth of the chalcocite enrichment in many of the Butte veins bears a definite relation to the depth of the zone of oxidation in the respective veins.

In an endeavor to solve the problem of the chalcocite formation, some investigations have been made. H. V. Winchell[18] succeeded in producing artificially, under normal conditions of temperature and pressure, chalcocite identical in chemical composition and physical character with the sooty chalcocite of the Butte veins. His experiments seem to prove conclusively that the formation of secondary chalcocite is easily possible under the conditions of temperature and pressure found in the upper levels of the Butte mines. Similar conclusions have been reached by Stokes and others in the laboratories of the U. S. Geological Survey.

Perhaps the most elaborate investigation of this subject was undertaken by Charles T. Kirk[19] who made careful chemical and petrographic analyses of the altered granite occurring in and along the Butte copper veins. He found that during the vein-forming processes the granite suffered great changes in chemical and physical character. These changes took place more or less gradually. He separated them into three general alteration phases, namely: (1) the chloritic phase, which marks the earliest stage of alteration; (2) the sericitic phase, marked by the development of great quantities of sericite through the further action of heated waters in (1); and (3) the kaolinitic phase, a change from the sericitic phase brought about through the action of descending sulphate waters or sericitized granite.

Phases (1) and (2), therefore, result from the action of deep-seated ascending waters; (3) is effected by the action of descending cold meteoric waters on phases (1) and (2). With these three alteration phases Kirk links certain generalized groups of minerals. He believes the early quartz-pyrite ores began to form with the early chloritic phases; that the copper mineralization during this and the succeeding

[18] Winchell, H. V., The Synthesis of Chalcocite and Its Genesis at Butte, *Engineering and Mining Journal*, vol. lxxv., No. 21, pp. 783 to 784 (May 23, 1903).

[19] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (Jan., 1912).

sericitic stage was principally enargite, bornite, and chalcopyrite; and lastly, that the chalcocite formation belongs entirely to the third or kaolinitic phase. He holds that kaolinite is wholly a product of cold meteoric water action and therefore the presence of it in deep levels indicates the presence of waters of meteoric origin. Many geologists, notably Gregory, dissent from this view and hold to the opinion that kaolinite may also be a product of ascending water alteration.

It is to be regretted that Kirk did not give a series of direct comparisons between samples of altered granite taken both from the sooty chalcocite zone and the deep levels. Certainly there is much yet to be learned concerning the relation between the altered granite and chalcocite formation. Even assuming for the moment that meteoric waters have sunk to great depth in the Butte veins, accompanied by the formation of kaolinite at all levels, it does not necessarily follow that the chalcocite was deposited from such descending waters. It cannot be doubted that the sulphate waters descended to depths greater than the lower limit of the sooty chalcocite zone, but it is evident that while the chemical effect of these waters upon the granite at greater depths may have been of the same general nature as in higher levels, that is, kaolinization, the chemical action toward copper mineralization was entirely different. The chalcocite of deeper levels is not necessarily a replacement of a sulphide mineral as in the upper levels. Since the replacement of the pyrite by chalcocite in the higher zones is accompanied by the formation of ferrous sulphate, the descending waters passing below the sooty chalcocite zone still retain an abundance of dissolved iron sulphates and possibly sulphuric acid to act on the seriticized granite, forming kaolin, as in the higher levels, but it is more than probable that the descending waters were entirely robbed of copper in the secondary chalcocite zone. The small amount of kaolinite present in the deeper levels as compared with the great abundance in the oxidized and sooty chalcocite zones indicates less activity, due either to dilution or to change in chemical composition of the solution.

From these considerations it is readily seen that the association of chalcocite with minor amounts of kaolin below the chalcocitization zone does not necessarily imply that both have resulted from the same solutions. The descending sulphate waters may still continue to form kaolin in regions of primary chalcocite after having deposited all of the copper burden in the region immediately below the zone of oxidation. It may not be difficult to understand meteoric waters reaching to unusual depths in the Butte veins, but the writer seriously doubts that such waters could retain copper in appreciable

quantities after moving downward for hundreds of feet in direct contact with newly formed chalcocite and an abundance of pyrite.

It is unfortunate that most of the granite samples used by Kirk in his investigations were collected from the Pittsmont vein, for the reason that this property is peculiarly situated with respect to the general topography of the district, and it is also in close proximity to the Continental fault, a fracture of comparatively recent occurrence. The effect of this fault upon the water level and oxidized zone may be readily understood by reference to Plate IV. It will be seen that the immediate effect has been to drop the former erosional surface and oxidized zone to a depth considerably below their former positions. The collar of the Pittsmont shaft at the present ground surface is about 400 ft. above the old erosional surface. In the mine workings the oxidized zone is from 250 to 300 ft. thick, measured downward from the former surface, and the sooty chalcocite belt is known to extend at least 500 ft. deeper. Summing up these figures, the result is reached that the deepest general working level of this mine (the 1,200 ft.) is not more than 600 ft. below the zone of oxidation, or, as a matter of fact, scarcely below the zone of sooty chalcocite. When it is remembered that in the Mountain View mine the sooty chalcocite zone is from 800 to 1,200 ft. thick, one is forced to the conclusion that Kirk's samples do not represent conditions far removed from the direct influence of copper-bearing surface waters. The results of his work are extremely interesting and of value, especially his investigations concerning the alteration of the granite, but in the opinion of the writer he has erred in attempting to apply his method of reasoning to the chalcocite of deep ore bodies of the Butte veins with which he is evidently unfamiliar. His results are valuable inasmuch as they further corroborate and establish, from a new point of attack, the conclusions already reached by others that the sooty chalcocite, and massive chalcocite to a limited extent, of the Butte deposits, have resulted from the work of downward-seeping sulphate waters whose copper was derived from the oxidized zone.

W. H. Weed[20] has set forth some facts which, in his opinion, tend to prove the secondary origin of the Butte chalcocite. He observes generally that the old quartz-pyrite veins were originally of very low grade and they became commercially valuable through the later addition of enargite, bornite, chalcocite, and other copper minerals. He believes that this copper mineralization followed various periods of faulting, the enargite and bornite being the first to appear, probably

[20] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, p. 152 (1912).

contemporaneous in a general way with the Blue and Steward fault system. Chalcocite, which forms the bonanza ores of the district, is thought by him to have been almost entirely a product of descending sulphide enrichment processes, acting at great depths, however, only where the older quartz-pyrite veins were crackled and broken by faults, thus permitting a ready passage for the downward-seeping waters. He cites many examples of such intersections of faults and older veins in support of this view, and maintains that the old quartz-pyrite veins are workable only where thus fractured.

The writer's own observations do not confirm Weed's conclusions as above outlined. Actual examination of a great many intersections of old quartz-pyrite veins by later faults has shown conclusively that as a general proposition the east-west veins are no richer at or near intersections with Blue vein faults than at other points along the vein except in cases where the fault vein ore shoots cross the older vein. It is extremely difficult to form even an approximate idea as to the extent of primary enrichment in the older veins due to the late faults of the Steward system. Mineralization processes were active in the early veins prior and subsequent to the Blue vein period, so that it is impossible to determine, in the absence of any characteristic minerals, what influence was exerted by the later faults upon the older veins. As might be expected, the fault vein intersections are usually accompanied by a breaking and shattering of both the older vein and the country rock in the immediate vicinity, thus developing favorable factors tending to greatly influence ore deposition at such points. In any case, where a chalcocite enrichment of a vein of the Anaconda system is shown to have resulted from the influence of an intersecting fissure of the Blue or Steward system there remains the strong probability that such enrichment is due to primary waters, if, as believed by the writer, the primary chalcocite was deposited in great quantities, after the appearance of these faults, not only within the faults themselves, but in the fractured older veins.

DISCUSSION.

L. C. GRATON, Cambridge, Mass.:—It has been my privilege to read with some care Mr. Sales's paper, and I feel it a sense of duty and a pleasure to discuss it briefly. After two months spent underground in the Butte mines by my associates—Messrs. Augustus Locke, A. M. Bateman, and E. H. Perry—and myself, it seems only fair to record our appreciation of the almost baffling structural complexity of the Butte veins, of the remarkable accuracy with which their mechanical details have been worked out, and of the unusually

effective and intelligent use of geologic conceptions in interpreting their subtler characters and significance. During this period, under the courteous and helpful guidance of either Mr. Sales, the members of his staff, or the geologists of other companies, we have been afforded opportunity to test many of the conclusions embodied in the present paper, and it is but just to state that as our observations have accumulated sufficiently the great majority of these conclusions have received unquestioned confirmation in our minds.

The pleasure consists in realization that the science of mining geology, to which so many of us have devoted ourselves, actually embraces the possibilities that this paper exhibits. The science of geology has been favored by many brilliant contributions of general nature, and by a smaller number of studies remarkable because of their detail and precision. I know, however, of no other piece of work like this, which, including Mr. Winchell's administration, virtually represents the combined results of over 15 years of work by a staff of trained geologists in a district which measures not much more from end to end than it now does from top to bottom. That this concentration of effort and observation has been required, and that it has been repaid by the results attained, are plentifully evident in Mr. Sales's paper. If one were inclined to doubt this after perusal of the text, surely he would be convinced by the maps and sections, several of which, I venture to say, are unapproached in detail and accuracy by anything attempted before.

Among the features of more general geologic significance mentioned by Mr. Sales that we have had opportunity to confirm to our own satisfaction may be noted the peculiar character of the Mountain View fault breccia; the essential unity of the period of primary mineralization in the district, regardless of the trend or age of the fractures; and the rudely concentric zoning exhibited both by intensity of alteration and by character of ore minerals deposited.

The subject upon which Mr. Sales places most emphasis, viz., the origin of the deep-level chalcocite, is one that possesses far more than ordinary interest for the four of us who are beginning here a country-wide study of secondary enrichment of copper deposits. I may say that Mr. Sales and I are not wholly in accord regarding some features of chalcocite occurrence and significance at Butte, but as my own views are not fully shared by my associates, it would seem the better part of valor to postpone any statement upon this subject until more thorough study can be given to it. That this problem is extremely complex will probably be admitted by all acquainted with it. In any event, after microscopic study of nearly a thousand specimens

of Butte ores, we are fully agreed with Mr. Sales that there are great quantities of original or primary chalcocite in these ores, and that several other rich copper minerals so plentiful here, notably enargite and bornite, are predominantly primary and not due in any important degree to influences active at or near the surface.

It appears, therefore, as pointed out elsewhere,[21] that the history of this greatest of the world's copper camps is not soon to be ingloriously terminated by a giving out of downward enrichment, but instead that, regardless of whatever alterations may have affected the deeper ores now known, profitable mining will probably be able to continue as far as the physical limitations of depth and vagaries of primary ore deposition will permit.

One might have wished that Mr. Sales had not confined himself so strictly to Butte. Comparisons in various respects with other mining districts might serve as tying-in points to those not fortunate enough to see this camp for themselves; and it would seem that the statement regarding genesis, particularly the nature of the depositing solutions, might have been strengthened if citations had been made to other mining regions for which similar ideas have been held. It is, however, a sufficient task to record the geology of a single district so complex as Butte, and those who would combat Mr. Sales's hypotheses should remember that these have been devised to accord with an enormous store of hard facts gathered through 13 years of close personal observation. I feel greatly indebted to Mr. Sales for his contribution.

W. C. RALSTON, San Francisco, Cal.:—When you speak of the acidity of the inclosing granite—that is, the primary granite—what is meant? Do you refer to the per cent. of acid present, and also how does the acidity of the altered granite adjoining the veins compare with the normal unaltered granite? To me the line of demarcation would be rather indefinite, and this dividing line does not appear to be clearly defined.

MR. SALES:—"Acidity" when applied to rocks is a comparative term only and it refers to the relative amounts of silica present. The quartz-porphyry of Butte is more "acid" than the granite because it contains 4 or 5 per cent. more silica. The silica content of the altered granite associated with the vein is a widely variable constituent. Sometimes it is more siliceous and often less siliceous than the unaltered granite, depending upon the degree of alteration and many other factors. In general terms, the line of demarcation between

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[21] The Sulphide Ores of Copper, *Trans.*, xlv., 67 (1913).

altered and unaltered granite is well defined, though there may be a slow gradation between slightly altered and highly altered material.

MR. RALSTON :—Is it a highly acid rock?

MR. SALES :—The Butte granite runs about 64 per cent. silica.

MR. RALSTON :—Now, as to the altered granite, what percentage has it?

MR. SALES :—It is variable, sometimes higher, but often lower in silica than the normal rock.

J. W. RICHARDS, South Bethlehem, Pa. :—May I call attention to the fact that where chemistry touches the field of geology a knowledge of physical chemistry will greatly aid the geologist in solving his questions. It is a great thing to have a thousand million dollars' worth of copper, but to the real geologist there is more satisfaction in knowing how the copper got there. We are now possibly groping in the dark because of a lack of knowledge of the reactions of copper-bearing solutions under high pressures. I think that is one of the weak points of the paper, especially where it enters upon theory and speculation as to how the primary chalcocite was produced. I think we should go to the laboratory and get more experimental information as a preliminary and guide to our theory and speculation.

Applied Geology in the Butte Mines.

BY FRANK A. LINFORTH, BUTTE, MONT.

THE object of this paper is to present a brief outline of the methods of geologic mapping employed in the Geological Department of the Anaconda Copper Mining Co., at Butte, and to show by means of a few typical examples the practical nature of the results obtained. The extremely complicated geological conditions early encountered in Butte mining led mine operators to realize the necessity of accurate detailed geologic mapping as an aid to successful development of the ore bodies.   The organization of the Anaconda Geological Department was undertaken in 1900 by H. V. Winchell, who, in conjunction with D. W. Brunton, worked out the essential methods of procedure which are, in the main, followed at the present time. Since the year 1906 the geologic work has been under the direction of Reno H. Sales, and in some respects the scope of the work has been slightly enlarged.   An excellent article describing the general equipment of the geological department was presented by D. W. Brunton at the British Columbia meeting of the Institute in 1905.[1] His idea, that data collected underground and recorded should be put to practical use and not remain a mere "inventory of the company's underground possessions," is the keynote to the success of applied geology in Butte.

The geologic notes are taken as soon as possible after the ground is broken so that any mistake in the mining may be corrected at once, or any particularly advantageous procedure may be suggested before any useless work is done.   Taking the notes underground is a comparatively simple matter, but a few necessary precautions may be pointed out.   The essential to success is that the notes shall show exactly what geological facts are disclosed.   The observer must discriminate carefully between important and unimportant exposures in making the record, especially as regards the relation between veins and stringers, between faults and minor slips, or between the characteristics of veins of different ages.   A simple color scheme has been adopted for making this record.   Red pencils are used to indicate

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[1] *Trans.*, xxxvi., 508 (1905).

vein filling, which may be ore, barren pyrite, in fact any metallic minerals or quartz, and the record of the minerals present is found in the written notes. Blue coloring indicates evidence of faulting, either as the definite planes of movement or the crushed granite resulting from such movements. The sketches are always supplemented with copious notes as to the dip and strike, character of mineralization, and condition of the surrounding granite. A loose-leaf system has been adopted for the notes, which allows keeping them in the form of a card index. The scale used for the notes is 40 ft. to the inch, although 20 ft. to the inch is found to be a better scale in places where the detail is especially complicated.

In the office the principal working maps are drawn to a scale of 40 ft. to the inch. They are prepared on tracing cloth, one level to a sheet, and made to register perfectly with each other. The notes are platted on these sheets as soon as the surveys are posted, and the same color scheme is used as that for the original notes. After each platting, these maps are studied in conjunction with all the recorded data of that particular vicinity, and ideas for future development or for alterations of the present plan suggest themselves. The details studied on these maps, which often lead to useful suggestions underground, are such matters as the position of a block of drag ore at the intersection of a fault and a vein; the sudden change in the character of a vein; the relation of dip and strike of a vein to those of a fault cutting it, and the correlation of faults, veins, and ore shoots from level to level.

A set of maps drawn to a scale of 100 ft. to the inch is also maintained in the office for each mine. This scale makes it possible to show a whole mine level on one sheet without having it inconveniently large, and many useful correlations of faults and veins in widely separated parts of the same mine are made with this set. All the details consistent with the smaller scale are platted on these maps, although the written notes are usually not transferred from the larger working maps. Each mine foreman and superintendent is provided with a copy of the 100-scale geologic maps of his mine and has them posted from the office set at regular intervals.

Next in importance are the vertical cross-sections, of which Fig. 1 is typical. They are made on a scale of 100 ft. to the inch and are taken at various intervals along lines depending on the strike of the veins to be studied. Since the veins and faults in this district present such a variety of strikes, it is obvious that no cross section can be prepared which will cut all of them at right angles. Therefore, the true dip for a vein whose line of strike meets the plane of the
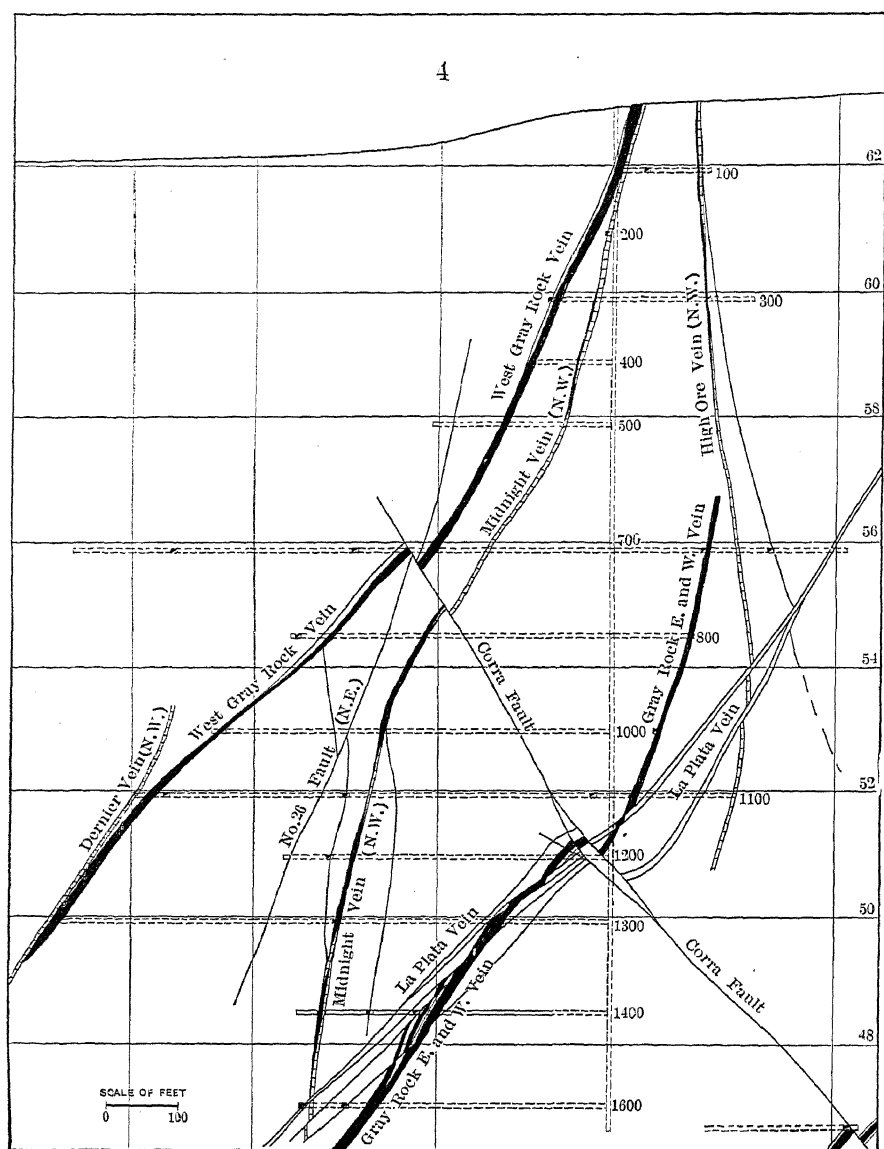
FIG. 1.—TYPICAL SECTION OF WEST GRAY ROCK MINE.

section at an acute angle cannot be used directly in making that section. In order that the geological relations on the section may be kept geometrically correct there is a convenient instrument in use which automatically reduces the true dip and angle of intersection to the angle of dip which must be platted. Vertical sections have frequently been instrumental in clearing up complicated geological structure, and many savings and discoveries have been made by studying them.

There is also maintained in the office a set of 200-scale tracings covering the entire district. Only the principal geologic features appear on them, but the proper relative importance of the veins and faults is maintained. This set of maps is particularly useful in correlating and identifying veins in widely separated mines.

Before presenting examples of a few of the typical discoveries made through the aid of geologic maps and sections, some of the numerous minor uses of geologic information should be pointed out.

In laying out a certain drain tunnel on one of the lower levels where only limited development work had been done, it was found that the Bell fault would be encountered somewhere between the points to be connected. It was further noticed that the drain tunnel would meet the fault planes at an acute angle, and, since the fault zone is 75 ft. wide, the tunnel would have to lie within it for a distance of 320 ft. It was therefore decided to run the tunnel to a point as near as possible to the Bell fault without cutting it, then to cross-cut the fault at right angles and continue the work in the footwall of the fault. The problem for the Geological Department, then, was to locate the fault on this partly developed level as nearly as possible, and to turn the tunnel in accordance with the determined position. Several cross-sections and projections were made from the known positions of the fault, and these checked fairly well for the desired position. When the course of the tunnel was finally turned, the Bell fault was cut at right angles only 15 ft. from the turn. This work will result in a minimum cost of timbering and maintenance of this drain tunnel.

The pump stations in Butte are exceptionally large underground excavations, and it is desirable to locate them in ground which is as free as possible from faults and fault veins. The Geological Department has been called upon to make examinations for this purpose, and in one case it was found advantageous to change the location of one of these pump stations. The position originally selected for it would have been crossed by certain faults and much difficulty would have been met both in the excavation and in future maintenance.

Where a raise was designed to connect two levels, it has frequently been possible to select a starting point such that the raise would be continuous in ore, whereas in any other position it would have to cross a fault before reaching the level above. In the West Gray Rock mine, it was estimated by projections that the faulted segment of a certain vein between two faults would only be a few feet in length on the 800-ft. level. The development work at the time was limited on this level, and the calculations had to be made from positions known on the 1,000-ft. level, where the faulted segment was considerably longer. In order to make the raise in ore continuously from the 1,000- to the 800-ft. level, it had to be placed in accordance with these calculations. The result was entirely successful, and the raise through this wedge-shaped block of ore reached the 800-ft. level at its very apex.

In another case, the geologic cross-sections and plans showed that the La Plata fault was nearly parallel in strike to an east-and-west vein. The dip of the vein, however, was steeper than that of the fault, so that in the stopes the fault generally encroached upon the work and presented all the appearance of a hanging-wall of the vein. The stope became narrower and the operators considered it a pinched place in the vein. The stope would probably never have reached the level above if the geologic conditions had not been understood. When the ore was finally cut off raises were continued from the top of the stope, but designed to cross through the fault and to make the fault a foot-wall rather than a hanging-wall. By this means, the ore above the fault was encountered at its lowest point and the stope continued to the level above. This procedure underground was suggested by studying that portion of the section, Fig. 1, which shows the La Plata fault and the Gray Rock east-and-west vein between the 1,200- and the 1,300-ft. levels. Fig. 1 is a cross-section through the West Gray Rock mine, and is typical of the 100-scale sets of sections carried for all the mines.

An important discovery made through a somewhat different agency is represented in Fig. 2. The main drawing is a portion of the 40-ft. scale map of the 1,800-ft. level of the Mountain Con mine, and the small sketch is a reproduction of the 20-ft. scale notes upon which information the discovery was made. It will be seen that without any geologic notes the drift No. 1824 would appear to be a drift on a continuous vein, and in fact it was stoped as such. The notes, however, displayed a significant difference in character between the westerly end, from which the work progressed, and the easterly end. The notation on the map indicates this difference. The westerly

portion exhibited the character of a quartz-pyrite vein, which it was known to be, but as the drifting continued the exposures were characteristic of the northwest fault vein system, notwithstanding the apparent continuity of the vein. This led to an investigation of the stope and the small sketch here reproduced was made. It showed these differences clearly to the geologist, and also the departure from the stope of the clay portions of the fault. Accordingly, the crosscut No. 1869 was run, and it encountered the faulted portion of the quartz-pyrite vein now being mined in drift No. 1828. As this drift
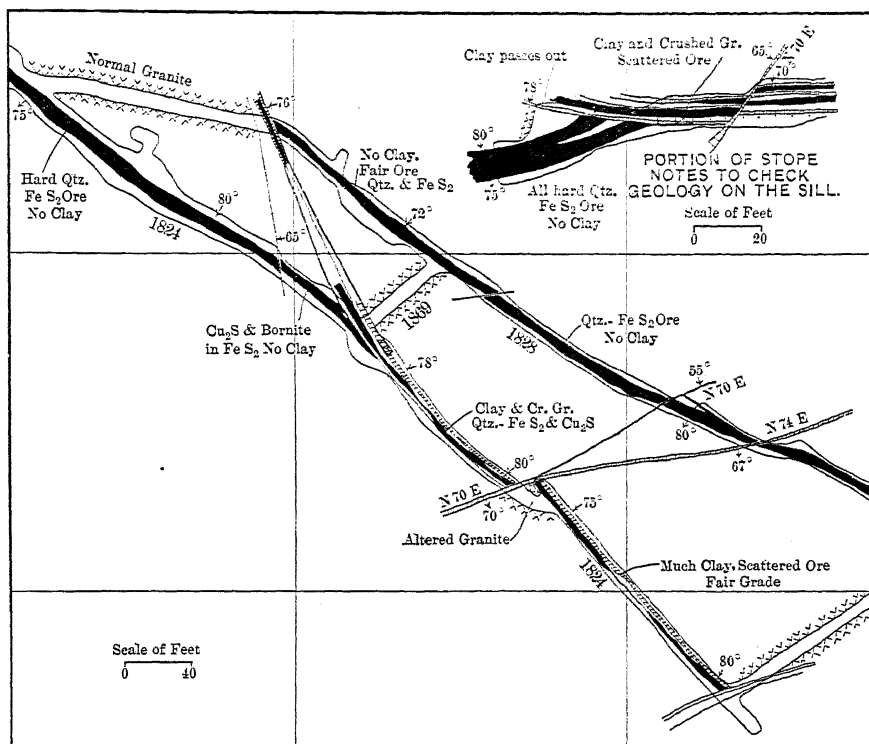
FIG. 2.—PORTION OF 1,800-FT. LEVEL, MOUNTAIN CON MINE.

was advanced westerly it intersected the fault vein as shown, and proved the correctness of the theory. After this discovery, the development of several other levels became merely a matter of projection.

Fig. 3 represents a portion of the 40-ft. scale maps of the 1,100- and 1,200-ft. levels of the West Gray Rock mine. It will be seen that the south-dipping east-and-west vein on the 1,100-ft. level lies north of the north-dipping Corra fault, while on the 1,200-ft. level
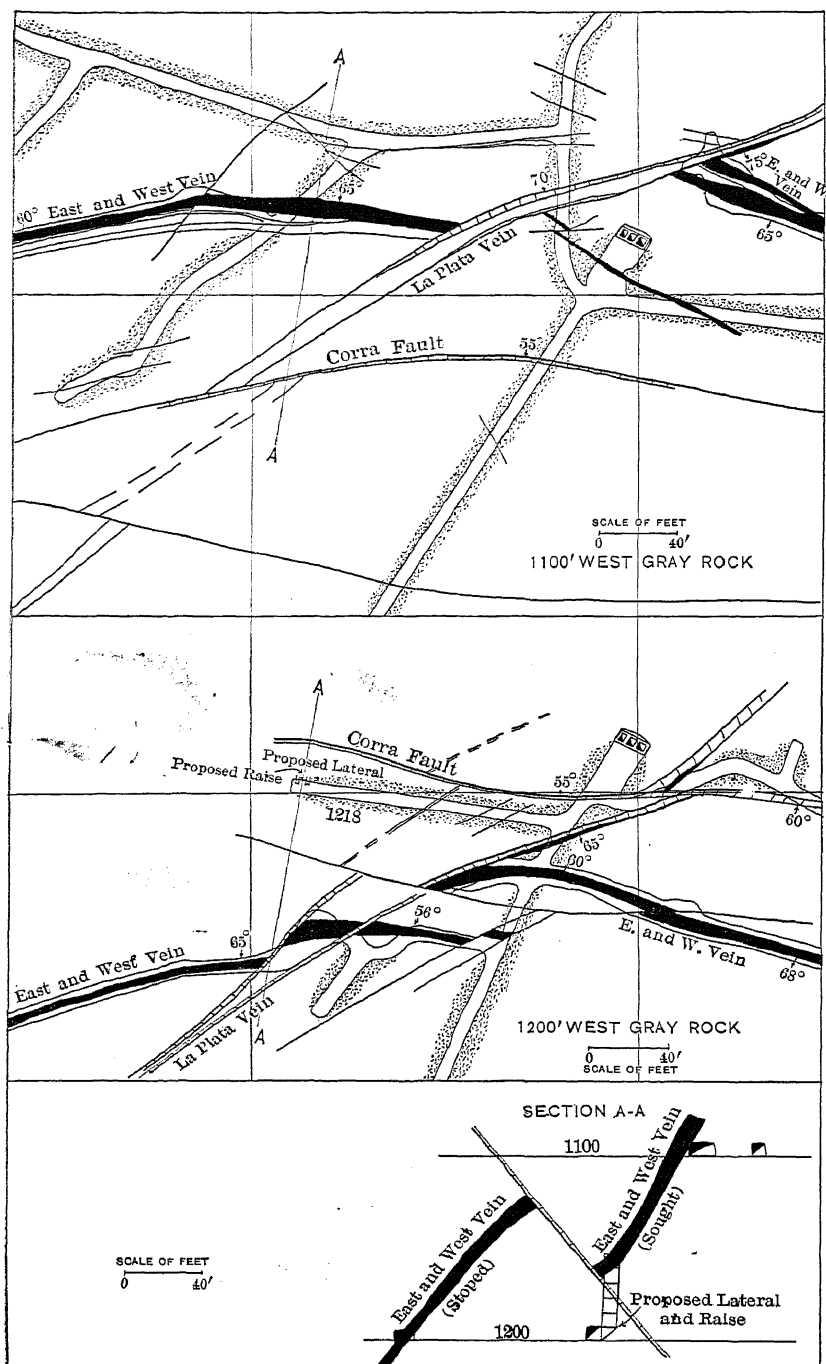
FIG. 3.—PORTION OF 1,100- AND 1,200-FT. LEVELS, WEST GRAY ROCK MINE.

these positions are reversed. The cross-sections clearly showed a displacement of the vein by the fault. The ore above the 1,100-ft. level was stoped to the 1,000 continuously in good ore and a stope was being worked above the 1,200, but, as shown on the maps, this stope could not reach the 1,100 on account of the fault. The ore on the 1,100 did not come down to the 1,200, for the same reason. The small section A–A shows these relations. It was therefore required to determine a point on the 1,200 from which a vertical raise could be run to intersect the vein as nearly as possible at its line of intersection with the fault. In other words, a vertical raise was to be so located as to reach the very bottom of the ore below the 1,100. A large-scale cross-section was drawn, making due allowance for the drag ore near the fault, and a course was given to the surveyor for the lateral drift No. 1218 in accordance with this section. The raise was to have one offset from the sill and to be vertical. The result of the work is shown in the small section in Fig. 3. Practically the bottom of the ore was encountered in this raise.

In Fig. 4 there is reproduced a portion of the 100-ft. scale maps of three levels of the High Ore mine, also a portion of the 40-ft. scale notes which led to the proper development of these levels. The map marked 300 Modoc shows a wide vein of good ore known as the Modoc vein. It was supposed that the entire drift was on this vein until the geologic notes showed that such was not the case. The portion near the shaft was barren; and the dip shown there was opposite to that at the east end. Bands of fault clay passed out of the drift as shown in the notes. It was decided that there were two veins exposed in this drift. At this time the drift on the level below (700 High Ore, see Fig. 4) had been abandoned at the point C. By carefully projecting the Modoc vein with its south dip from the 300 Modoc and also the other vein which is called the Edith May with its north dip, it was found that the line of intersection of these two veins would cross the level of the 700 High Ore at a point beyond the face of the old drift. Drifting was therefore continued in order to reach this determined point, and a certain amount of the north wall of the Edith May vein was broken so as to see the ground north of it. Within a few feet of the predicted intersection the Modoc vein was found and was stoped continuously to the level above. The next move naturally was to get the faulted portion south of the Edith May vein, which was accomplished as shown on the map. This information could now be extended to the next level below. The development work on the 900 High Ore had only reached the point D, Fig. 4, but the line of intersection of the Modoc vein and the Edith
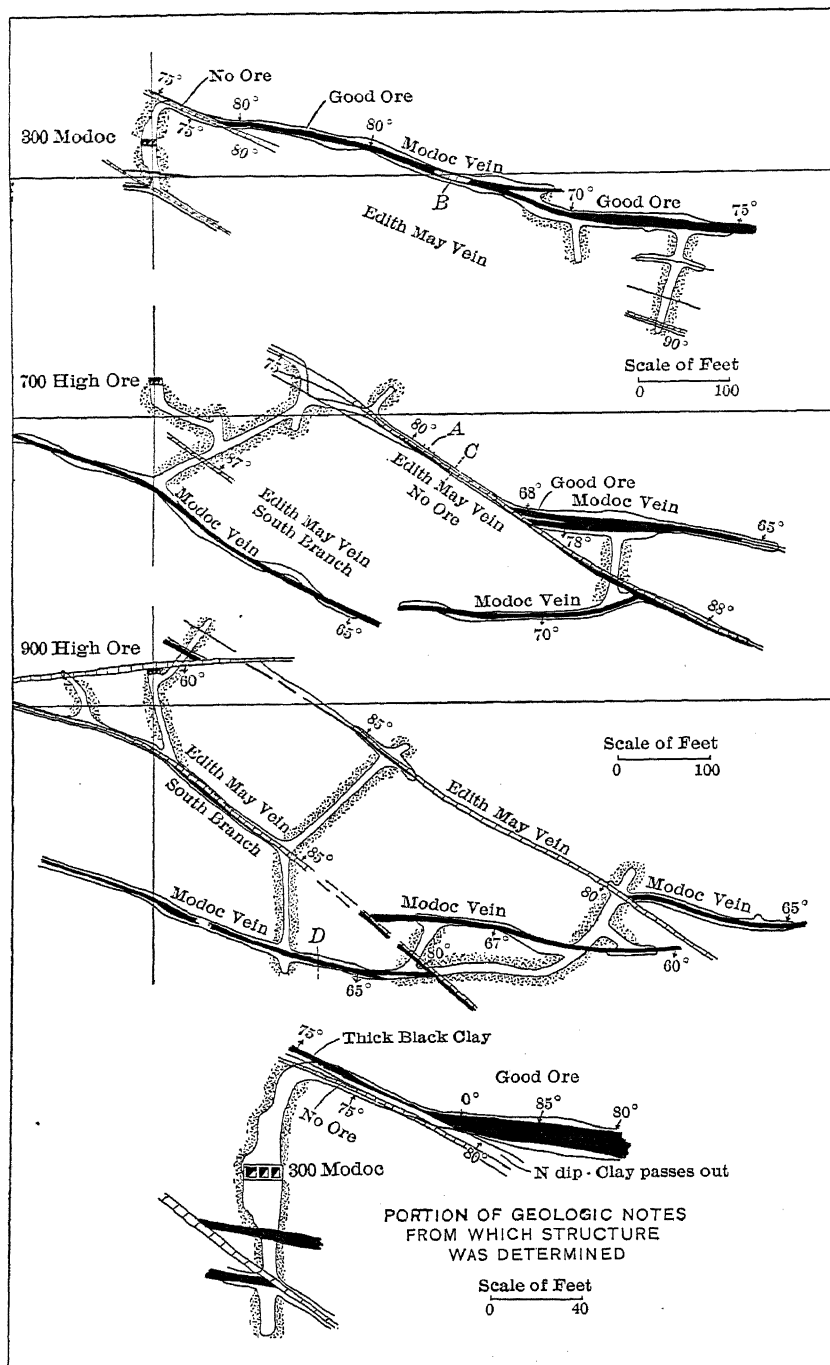
FIG. 4.—SECTION OF THREE LEVELS OF THE HIGH ORE MINE.

May vein was now well known on two levels and in the stopes, so it could be carried to the 900 with considerable accuracy. The underground work was therefore continued on the 900 with this point in view, and the map shows the accuracy with which the intersection was located. Nearly all of the ore thus developed on these three levels has been stoped.
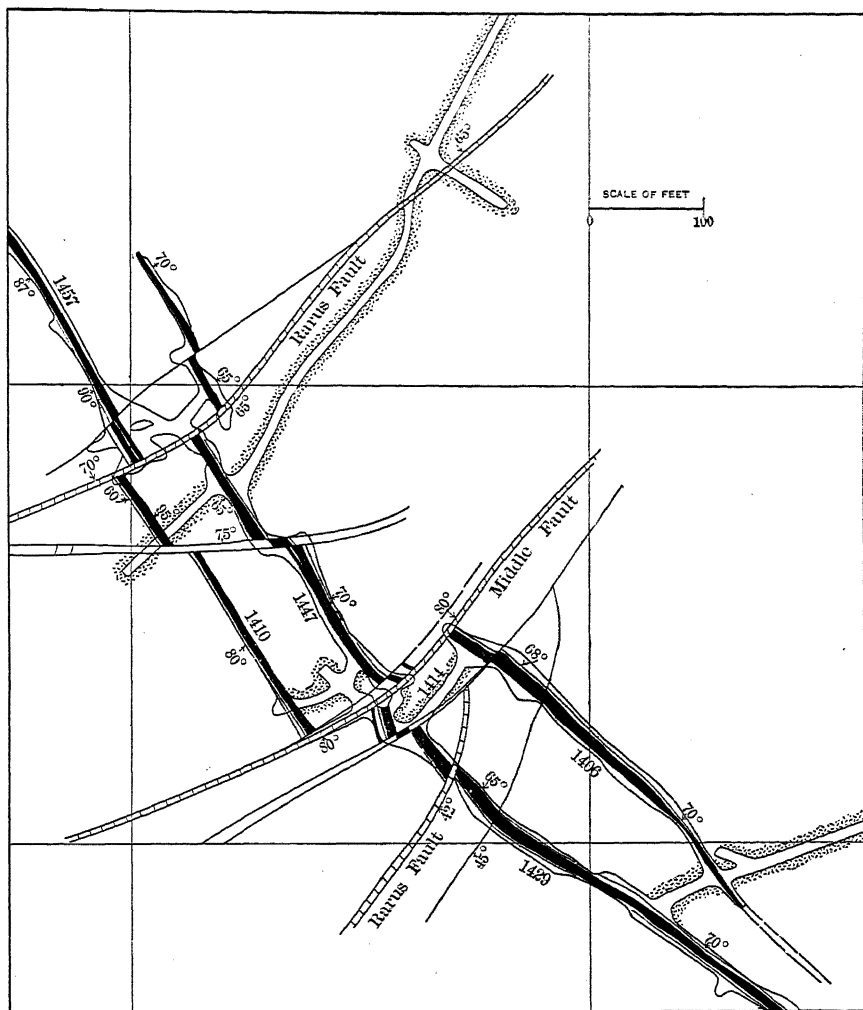
FIG. 5.—PORTION OF 1,400-FT. LEVEL, MOUNTAIN VIEW MINE.

In Fig. 5 is shown a portion of the 100-ft. scale map of the 1,400 level of the Mountain View mine. In order to understand the geologic work done in this case, the reader's attention must be called to drifts Nos. 1429 and 1447. The remaining numbered drifts

had not been run until the ore in them was discovered through the application of geologic knowledge of the conditions. The veins disclosed in these first drifts were considered to be faulted segments of the same vein. Later, however, in correlating with other parts of the mine, the Middle Fault (see Fig. 5) was identified, but the direction of the throw on this fault was known to be to the left instead of to the right, as the former interpretation would require. Therefore, the drift No. 1447 must contain the faulted portion of a vein lying north of No. 1429, and further, the faulted segment of No. 1429 must lie south of No. 1447. The map shows that this reasoning was correct, as the missing portions of the vein were found in drifts Nos. 1406 and 1410, respectively. The next step was to reach the south vein above the Rarus Fault (see Fig. 5), which was done in drift No. 1457. Thus everything was accounted for, and three valuable ore bodies were won.

The examples of geologic work given above are typical of the work carried on in the Butte mines, although many more might be cited. Of course, all of the suggestions made by the Geological Department do not result in the discovery of ore bodies, but a failure to find the vein predicted is unusual. When the underground work reaches a vein as predicted, but cuts it in a barren place, calculations on the pitch of the ore shoot in the plane of the vein are made, and the downward extensions of such shoots have been located.

Graphic determinations of the direction and amount of movement in the planes of the principal faults have been made, and the knowledge of these displacements, both vertically and horizontally, is very helpful in locating faulted veins, or identifying faulted segments of veins. It has been recognized that the amount of horizontal displacement of several veins by any particular fault may vary considerably, and this variation has been studied graphically. The relation between the amount of horizontal displacement, on one hand, to the dips of the fault and vein and the angle between their strikes, on the other hand, has been studied, and has resulted in useful deductions. In order to show the variety of horizontal displacements possible on one fault, owing to differences in angles of dip and angles of intersection, Fig. 6 has been prepared. The four diagrams represent the four possible cases, namely:

A. Dip of fault steeper than dip of vein, but in opposite direction.

B. Dip of fault flatter than dip of vein, but in opposite direction.

C. Dip of fault flatter than dip of vein, but in same direction.

D. Dip of fault steeper than dip of vein, but in same direction.

In all cases a normal downward displacement of 100 ft. is assumed
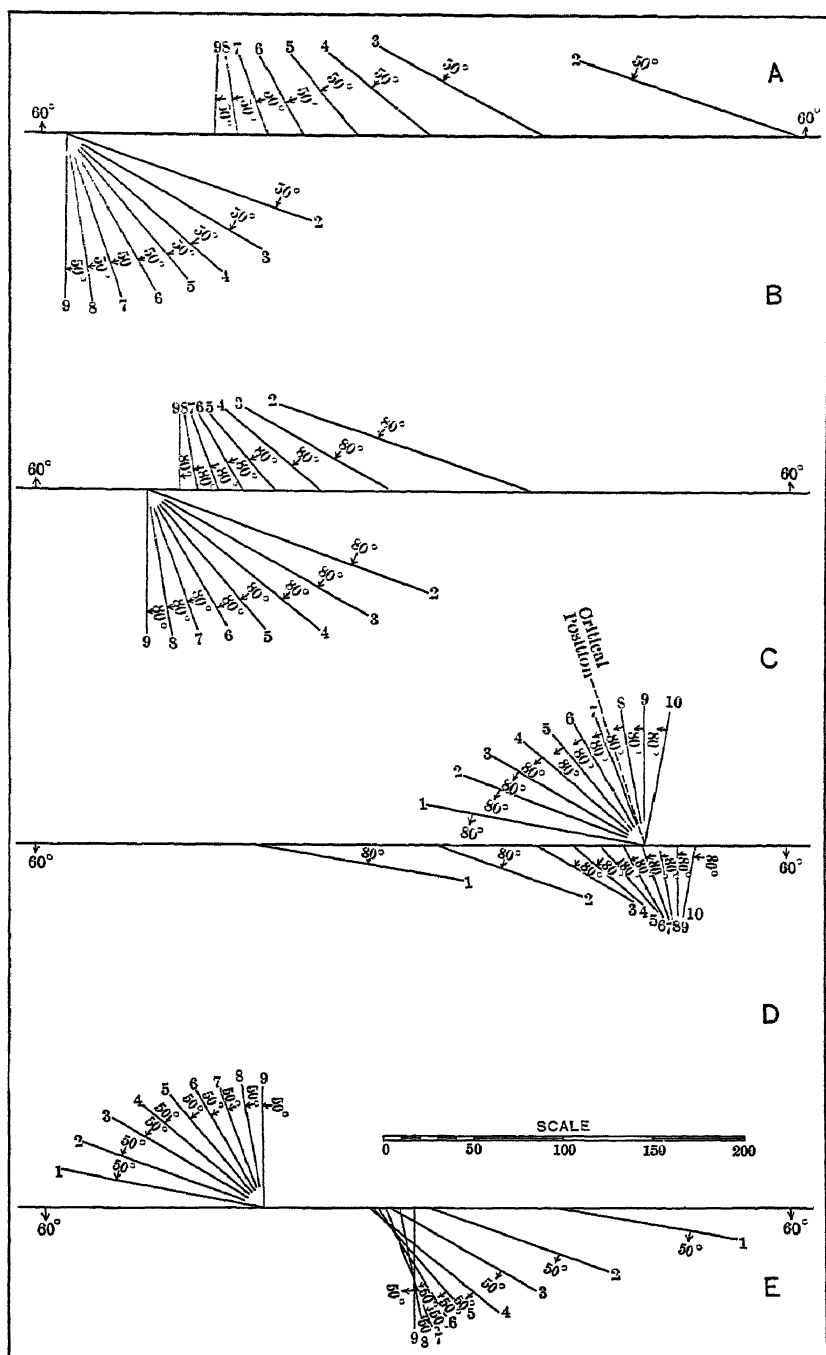
FIG. 6.—HORIZONTAL DISPLACEMENT OF VEINS BY FAULTING.

and the veins meet the fault at angles of from 10° to 90°. The various directions and amounts of horizontal displacement are shown. In one case it will be seen that the horizontal throw increases positively in direction to a maximum, and then becomes negative. The angle of intersection of the vein with the fault that gives the turning point is called the critical angle, and a means of determining it graphically for any set of dips has been devised. It must be understood, however, that these graphical determinations have their limitations and must be applied accordingly.

Although this paper has dealt largely with problems related to structural matters in the Butte district, much attention is given to the study of the complicated mineralogical and chemical problems involved, but they fall beyond the scope of this paper.

DISCUSSION.

B. H. DUNSHEE, Butte, Mont. :—I merely want to testify to the value of the Geological Department so far as the Anaconda Copper Mining Co. is concerned. The information that the mining men get from the Geological Department is exceedingly valuable. Many times in mining underground we come to a question that is obscure; we don't know exactly where we are or what to do. Instead of going at it blindly and perhaps wasting time and money searching for the continuation of the vein, we merely go to the Geological Department, and the geologists come to our assistance, and almost every time their conclusions are correct; the work they do is intelligent, and again I want to testify to the value of the accurate maps of the veins and fault systems as worked out by our Geological Department.

Mineral Associations at Butte, Mont.

BY D. C. BARD AND M. H. GIDEL, BUTTE, MONT.

(Butte Meeting, August, 1913.)

THESE notes are based on the megascopic study of a suite of 2,400 specimens of minerals and ores from the Butte mines, combined with field observations at intervals over a period of several years.

The estimate of average vein composition is based on field acquaintance with the veins. It is probably correct in its broader proportions, and should be of use for comparison.

Chemical and Mineral Composition of Unoxidized Vein Matter.

Per Cent.

Oxygen, ... 48.00—Quartz, rhodonite, rhodochrosite, sericite, kaolinite, barite, calcite, and hübnerite.

Silicon, ... 36.00—Quartz, rhodonite, sericite, kaolinite, and willemite (very rare).

Sulphur, ... 5.00—Sulphides of iron, zinc, copper, lead, and sulphates of barium and rarely calcium.

Iron, ... 5.00—Ferric sulphides (pyrite, bornite, chalcopyrite), also in sphalerite, tetrahedrite, and rarely hematite and siderite.

Manganese, ... 2.00—Rhodonite, rhodochrosite, and hübnerite.

Zinc, ... 1.00—Sphalerite, very rare wurtzite and willemite, and tetrahedrite and tennantite.

Carbon, ... 0.50—Rhodochrosite and calcite.

Copper, ... 0.75—Cuprous (+) and cupric (—) sulphides.

Aluminum, ... 0.50—Sericite and kaolinite.

Hydrogen, ... 0.50—Water, sericite, and kaolinite.

Potassium, ... 0.20—Sericite.

Lead, ... 0.20—Galena.

Arsenic, ... 0.10—Tennantite and enargite.

Barium, ... 0.10—Barite.

Calcium, ... 0.10—Fluorite, calcite, and rare gypsum.

Antimony, ... 0.02—Tetrahedrite, famatinite, and silver sulphantimonites.

Fluorine, ... 0.05—Fluorite.

Silver, ...	—Argentite, and sulphantimonites.
Tungsten, ...	—Hübnerite.
Phosphorus, . . .	—Wavellite.
Bismuth, ...	0.05—Sulphide (?)
Tellurium, . . .	—Telluride (?)
Selenium, ...	—Selenide (?)
Cadmium, ...	—Sulphide (?)
Gold, ...	—Native and telluride (?)

The area considered in these notes is limited by the Blue Bird mine on the west, Walkerville on the north, the Pittsmont mine on the east, and Silver Bow creek on the south.

Only such minerals are considered here as are thought to have some bearing on the genesis of the ore deposits. The oxidized zone minerals are in general omitted.

Mineral Composition in Relation to Chemical Elements in the Vein Matter.

Iron is deposited entirely as ferric sulphides.

Manganese is deposited as carbonate and silicate, rather than as a sulphide, possibly because of insufficient sulphur to satisfy all the bases.

Zinc is deposited solely as sphalerite except very rarely for some willemite. The absence of appreciable willemite suggests sufficient sulphur to satisfy the zinc. Wurtzite has been found in the upper levels of the Gagnon mine.

Copper is usually deposited as a cuprous compound, rarely as the cupric covellite.

Arsenic is deposited solely as the sulpharsenate, enargite, and the sulpharsenite, tennantite. Arsenopyrite is absent, suggesting a stronger affinity of the arsenic for copper than for iron. Also simple arsenic sulphides are absent, suggesting that in the presence of excess of copper only the sulpharsenic compounds are formed. Arsenates are lacking in the oxidized zone.

Aluminum occurs in sericite and kaolinite, which formed in place from the feldspars of the granite walls. The alteration of the feldspars to sericite freed an excess of silica, which occurs as quartz disseminated through the sericite in addition to that originally indigenous to the granite. Rarely, aluminum has been deposited along fractures in the vein matter as wavellite.

Water in the vein minerals is in very little if any greater quantity than that in the original granite.

Lead is found only as galena, the sulpho-salts of lead being absent.

Calcium occurs as fluorite, apparently being one of the earlier, high-temperature minerals. Gypsum rarely occurs in delicate crystals in vugs. Calcite is not common.

Antimony occurs in the sulphantimonites of copper and silver—never as stibnite.

Tungsten is found solely in hübnerite, usually in vugs.

Bismuth, tellurium, and selenium have been noted in the smelting operations, but have not been recognized in minerals. One specimen

of argentiferous chalcocite from the Gagnon mine assayed 0.5 per cent. of bismuth.

Source of the Vein Materials.

Most of the vein minerals contain elements foreign to the granite country rock, at least in the concentrated form found in the veins. Exceptions to this statement are the sericite and kaolinite which could result from the alteration of the granite in place; and the wavellite which is locally concentrated, but no greater in total amount than could be supplied by the phosphorus in the apatite of the granite. The original granite contained more than sufficient calcium and barium in feldspars to satisfy the fluorite, calcite, gypsum, and barite of the veins.

Likewise, the wall rock could and probably did furnish 50 per cent. of the quartz of the veins. Also there is sufficient iron in the original granite to satisfy the average iron content of the veins, although the iron was subjected to concentration and alteration to the ferric condition. It should be noted, however, that our estimate of average iron content may be too low.

The elements which were foreign to the granite wall rock in any such amounts as are found in the veins are: Silicon, sulphur, manganese, zinc, carbon, copper, arsenic, lead, antimony, silver, tungsten, bismuth, selenium, tellurium, cadmium, and gold. To this list should be added oxygen, although there is nearly the same amount of oxygen in the granite as in the veins.

The elements which are found in the veins in no greater quantity than in the original granite are: Iron, aluminum, hydrogen, potassium, barium, calcium, fluorine, and phosphorus.

The elements which are found in greater quantity in the granite than in the veins, and which therefore have been in part or entirely removed by the vein-forming processes, are, in order of amount: Aluminum, calcium, sodium, magnesium, potassium, phosphorus, and titanium.

Observed Paragenesis of the Vein Minerals.

Quartz is found in all relations to the other minerals, being deposited throughout the mineralizing period. The only exceptions are that no quartz has been observed later than calcite or wavellite.

Fluorite was among the first minerals formed. It is earlier than rhodochrosite and sphalerite. But that it was forming over some length of time is shown by its occurrence in the east-west veins and in the later northwest-southeast veins.

Another early vein mineral is hübnerite. It occurs usually in vugs or in porous parts of the veins, associated with quartz. It is sometimes replaced by pyrite, chalcopyrite, and enargite.

Molybdenite is sometimes given as a vein mineral from Butte, but has been observed by us only in the earlier pegmatite and aplite dikes which are not structurally connected with the vein systems. One occurrence of wulfenite has been noted in the oxidized zone associated with wavellite.

Rhodonite and rhodochrosite are found intimately intergrown and associated with quartz, sphalerite, and rarely with willemite. They have been noted later than some sphalerite, pyrite, galena, and chalcopyrite; also earlier than some sphalerite. Rhodochrosite earlier and later than bornite has been observed. The only evidence that some of the rhodochrosite may be later than the rhodonite is that the rhodochrosite is more often found along the fractures and walls of the veins. The evidence is not clear that rhodochrosite is an alteration product of rhodonite. We have no evidence that rhodonite becomes, with depth, more plentiful than rhodochrosite. They both appear to be primary minerals contemporaneously deposited. It seems to be a fact that the manganese minerals are less plentiful in the deeper parts of the veins. The manganese minerals are rare in the copper veins.

Barite is a primary mineral, widely distributed but in small quantity. It is sometimes syngenetic with the sulphide minerals, more especially the sulpho-salts, and sometimes later than them. One specimen shows barite with later quartz and tennantite.

Gypsum is very rare. It has been observed in long, delicate crystals in vugs in the deeper levels, and is evidently primary. It is too rare to have any paragenetic significance. Gypsum, secondary from fluorite, occurs in the oxidized zone.

Calcite is widespread, but not common. It is apparently among the last minerals to form.

Wavellite is a rare mineral found in crusts in older vein filling. No mineral has been observed later than it.

Vivianite occurs rarely in sheaf-like crystals in vugs. It is apparently a late mineral.

Pyrite is of all ages.

Marcasite is not common. Its distribution does not seem to depend on depth, as it has been found on the 1800-ft. level of the Badger vein. Here it seems to be later than the underlying pyrite-bornite-chalcocite ore of the specimen.

Covellite in Butte is widespread in the copper veins, but is not plentiful. Certain veins carry larger amounts of it. It is generally crystalline and primary, being earlier or at least syngenetic with enargite and pyrite. One specimen shows covellite veinlets in enargite. It often alters to chalcocite, enargite, and chalcopyrite.

Enargite is one of the earlier vein minerals. In some cases it alters to tennantite. It frequently alters to chalcopyrite, and is also replaced by bornite. It is often syngenetic with bornite and chalcocite.

Chalcopyrite is not a common primary mineral. It is found syngenetic with bornite and pyrite. In the copper veins it occasionally replaces enargite, chalcocite, covellite, bornite, and sphalerite. Chalcopyrite has been found deposited on pyrite. Tennantite deposited on chalcopyrite has also been observed.

Tennantite occurs as a late-formed mineral, particularly on the edges of the copper zone. It is replaced by chalcopyrite, and replaces sphalerite and galena.

Chalcocite is generally a primary mineral syngenetic with bornite, enargite, and pyrite. It also occurs later than these and replacing them. It also replaces sphalerite at times.

Tetrahedrite is not common. It appears to be an early, primary mineral, sometimes syngenetic with sphalerite. It is replaced by chalcopyrite.

Hematite has been found in one good specimen from the 2,200-ft. level of the Anaconda mine. It is associated with chalcocite, pyrite, and quartz, and appears to be syngenetic with them. It is rare.

Galena is common in the silver zone. It is syngenetic with sphalerite, chalcopyrite, and pyrite.

Sphalerite is common in the silver zone. It and galena have not been observed later than copper minerals except in one specimen of sphalerite in a vug from the 2,200-ft. level of the Speculator mine, which is inclosed by quartz, bornite, and enargite.

The mineralogical relations in the Butte veins do not suggest distinct and separate periods of mineralization, but rather one continuous period with varying degrees of intensity. But a small portion of the copper in the Butte veins seems to result from enrichment by descending, meteoric waters.

The Southern Cross Mine, Georgetown, Mont.

BY PAUL BILLINGSLEY, BUTTE, MONT.

Introduction.

THE Georgetown mining district is located in Deerlodge county, Mont., about 20 miles west of Anaconda. It lies along the divide between the headwaters of Warm Springs creek, draining eastward, and Flint creek, flowing west and north through the Philipsburg valley.

The actual divide is formed by Cable mountain, a narrow ridge running north and south, terminating to the south in low hills overlooking George-town lake, and to the north in a high, rugged plateau that falls off to the west to the wide valley of Flint creek. On the eastern, or Warm Springs, slope of Cable mountain is the Cable mine, with the Hidden Lake mine a few miles north. On the western slope is the Southern Cross group, with smaller mines, the Twilight, Cincinnati, Red Lion, and Big Bill, at intervals to the northward. The Georgetown mines lie in the hills south of Cable mountain; the Philipsburg mines, of which the Granite Mountain, Bimetallic, Hope, and Trout are the chief, form a separate group to the north, overlooking Flint Creek valley and Philipsburg. (See Fig. 1.)

The region ranges in elevation from 5,000 to 9,000 ft. It is, with the exception of the lower valleys, well timbered with small fir and pine, although Cable mountain and the higher summits to the northward rise slightly above timber line. The main valleys have the smooth contours of glaciation, and their mouths are frequently choked with morainal deposits.

General Geology.

In a very general way the valley of Flint creek coincides in this region with the boundary between the pre-Cambrian slates of western Montana and the folded Palæozoic rocks of the Rocky Mountain system. The great limestone formations of the Carboniferous and Devonian, lying at flat dips, extend from Anaconda well toward the head of Warm Springs creek, but on the divide the Cambrian appears, closely folded and intruded

by irregular masses of granite. Cable mountain itself is an anticline of the basal Cambrian quartzite and underlying shales, flanked by steeply dipping limestones. The strike of the beds coincides with the north-south direction of the ridge. On the south the fold is terminated by the small granite mass that extends from Cable to Georgetown, and on the north abuts against the edge of the larger Philipsburg batholith. The distance between these granite areas is about 7 miles; the width of the anticline about 2 miles. In this limited area the Cable, Red Lion, and Southern Cross groups are located.

The ore deposits of the district may be conveniently classified into four types: 1. Fissure veins in granite; 2. Contact metamorphic deposits; 3. Fissure veins in limestone; and 4. Replacement bodies in limestone.

The Granite-Bimetallic mines are, of course, the chief representative of the first class, and the Cable mine is an equally well-known instance of the second. These mines have been described in detail by W. H. Emmons[1]. The third and fourth types overlap somewhat, and both may often be found in a single mine, but at their extremes they are represented by the Gold Coin, a quartz-filled fissure with no replacement of the walls, and by the Southern Cross, a characteristic replacement deposit in limestone.

History.

The Southern Cross claim was located in 1866, but this first location was allowed to lapse, and the ground was relocated by Salton Cameron several years later. Cameron developed the Southern Cross vein, and shipped some ore to East Helena. Stamp milling was first attempted in 1884, but the results were unsatisfactory. Under the management of a Butte company production increased, and by 1893, 30,000 tons, with a gross value of $750,000, had been shipped.

In 1904 the mine was leased by Lucian Eaves, who discovered a new ore body to the west of the old Southern Cross vein. This yielded more than $300,000 gross. In 1906 the property again changed hands. The New Southern Cross Mining Co. installed a wet mill and prosecuted vigorous development work. This resulted in the discovery of new ore bodies between those formerly exploited, and greatly increased the probable life of the mine. In 1910 the Anaconda Copper Mining Co. purchased the property, and in the year following built a railroad from Browns spur, 5 miles from Anaconda, to the mine. The ore, high in iron and desirable as a flux, is smelted direct at the Washoe Smelter, so that the two great difficulties of the past, the long wagon haul and the refractory milling character of the ore, need not be contended with in the future.

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[1] *Bulletin No. 315, U. S. Geological Survey (1907).*
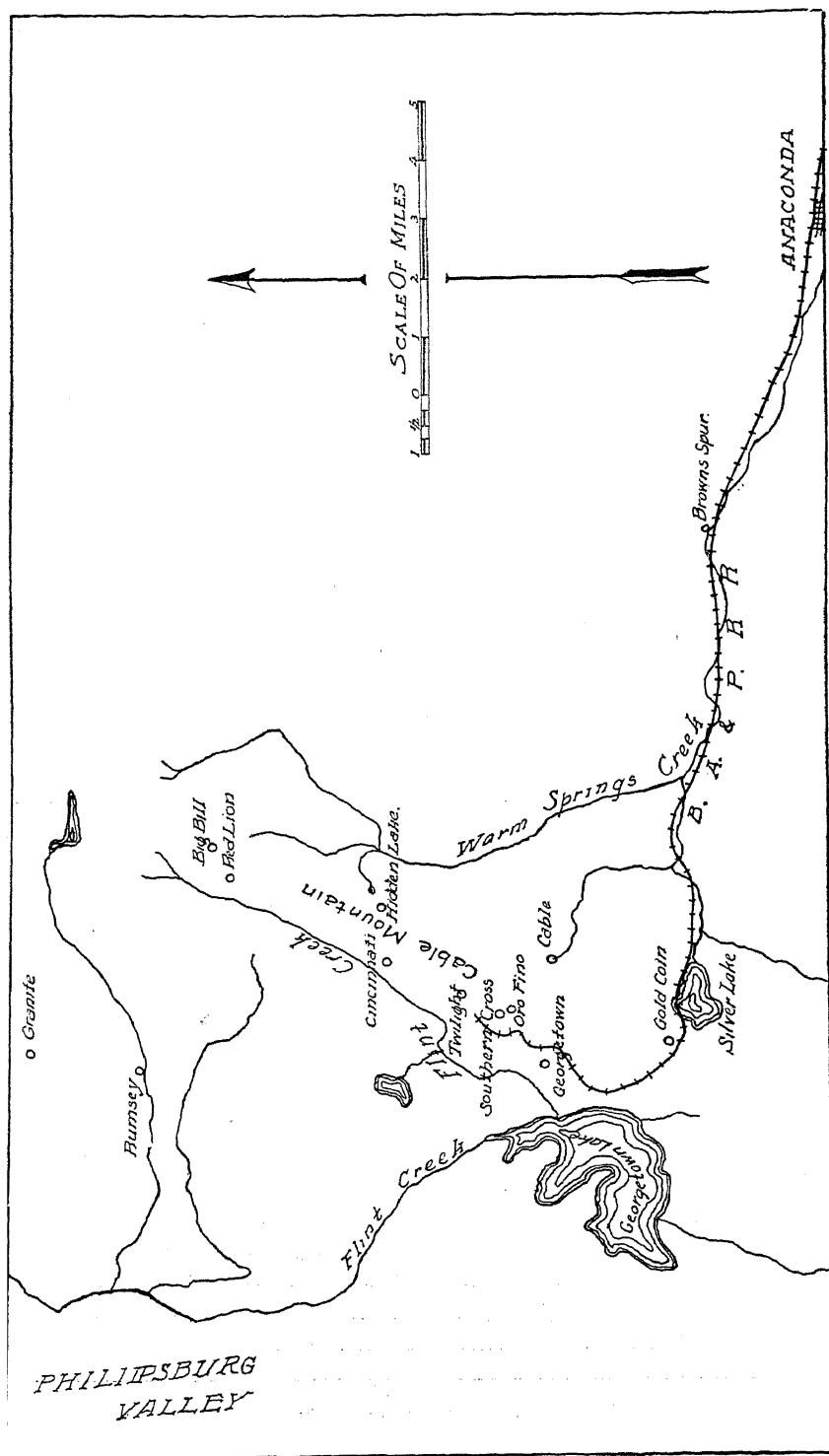
Fig. 1.—Index Map of the Georgetown District, Deerlodge County, Mont.

Geology.

The ore occurs as replacement bodies in limestone. The sedimentary succession in the district consists of the following members in ascending order:

Pre-Cambrian.. 1. Spokane shale.—Red and green shale.
                2. Flathead Quartzite.—White quartzite.
                3. Silver Hill Formation.—Calcareous shale.

Cambrian........ 4. Hasmark Formation.... { Lower magnesian limestone.
                                           Shale.
                                           Upper magnesian limestone.

                5. Red Lion Formation.... { Shale.
                                            Laminated limestone.

At the mine these strike slightly east of north, and dip from 50° to 60° to the west. The productive members of the series are the two magnesian limestones of the Hasmark formation. The Oro Fino ore body is near the base of the lower; the ore bodies thus far developed in the Southern Cross are in the upper, between the characteristic crinkly Red Lion limestone and the calcareous shale member of the Hasmark. Fig. 2 shows the general relation of the ore and the strata.

The limestones in the mine are not, in general, strongly metamorphosed. Certain favorable beds are locally changed to finely crystalline white marble, but the contact minerals, tremolite, garnet, etc., are conspicuously absent. On the surface the granite of the Cable batholith appears about 2,000 ft. south of the mine, cutting across the sediments at right angles. Underground, however, a small, badly decomposed dike in the southern part of the Southern Cross workings and a similar intrusion in the Oro Fino are the only indications of igneous activity.

Faulting likewise occupies a subordinate place in the geology of the mine. Certain of the ore bodies which cut across the bedding may occupy pre-mineral fractures; smaller fissures, with a strike of N. 80 W. and steep eastward dip, sometimes prominent as walls of the ore bodies, may indicate very slight post-mineral displacement; and a fault with the same strike and dip traverses the northeastern portion of the workings. The pre-mineral fractures, however, can rarely be followed for a score of feet beyond the vein filling, and the later faults have a displacement too slight to affect the ore bodies otherwise than by off-setting the walls into irregular steps. The limestone itself shows no extensive crushing or deformation.

The country rock at the Southern Cross, therefore, consists of massively bedded magnesian limestones, locally recrystallized but with little contact metamorphism, and comparatively undisturbed by faulting.
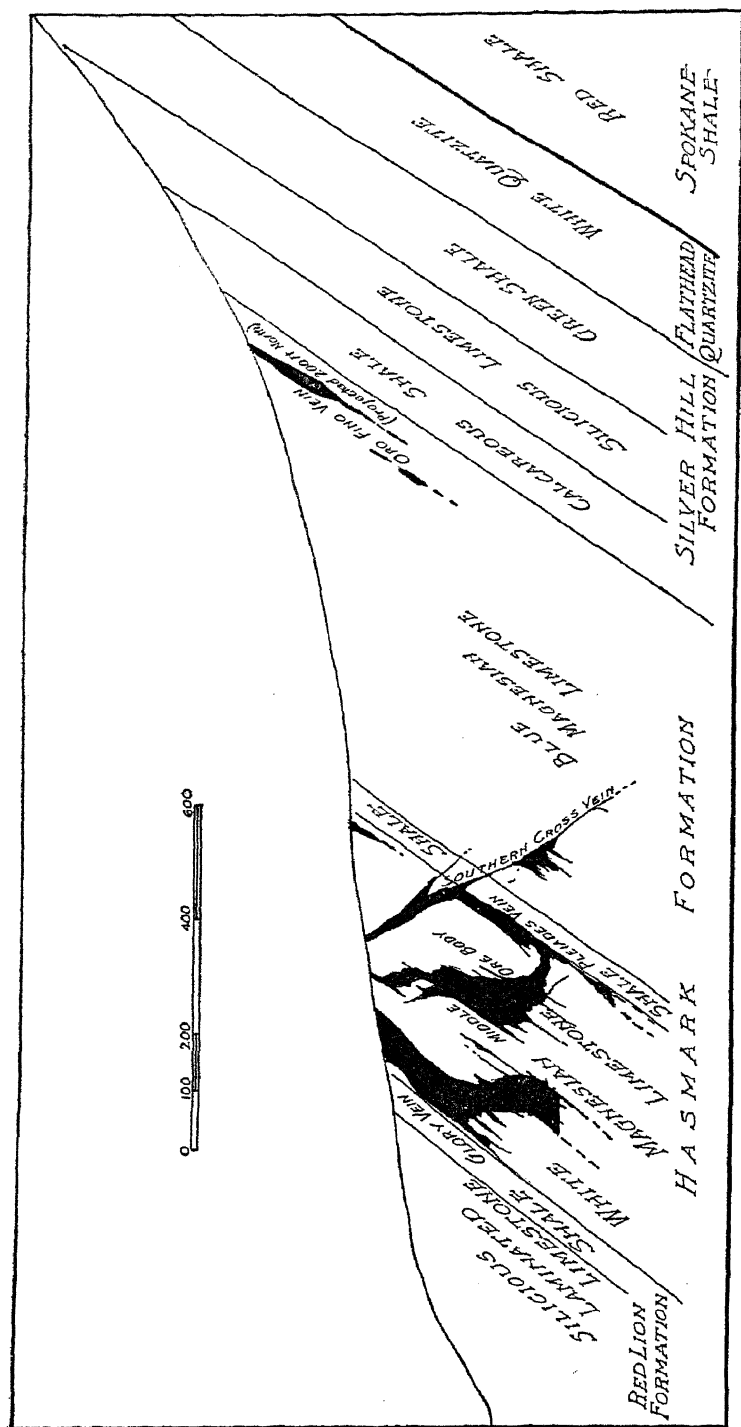
FIG. 2.—CROSS-SECTION OF THE SOUTHERN CROSS GROUP, SHOWING RELATION OF ORE BODIES TO BEDDING.

Character of Ore Bodies.

The ore consists of irregular bodies of soft gold-bearing iron oxides, replacing the limestone. The common minerals are limonite, hematite, siderite, and magnetite, with lesser amounts of quartz and calcite. The two first mentioned, with partly replaced earthy limestone, form the greater portion of the ore. Carbonates of copper are occasionally found, coating the iron minerals and concentrated in the porous portions of the limonite. A residual brown clay is common around the borders of the deposits.

Sulphide ore has not been encountered in the workings to the present depth of the mine—400 ft.—but residual bunches are occasionally found in the oxidized ore bodies. These bunches occur as pyrite cores, surrounded by massive hematite or limonite, and traversed by cracks and fissures of the partly altered sulphide. The entire amount of this material is slight, but the separate occurrences are so similar as to be strongly suggestive of the primary ore underlying the oxidized zone.

The uniform character of the limestone and the absence of strongly defined bedding planes have combined to give the deposits extreme irregularity of outline. With these factors reduced to a minimum, the extent of the replacement has been determined by such remote and obscure causes as local intensity of the mineralizing solutions, pre-mineral cleavage planes, and varying solubility of portions of the limestone. With the destruction of such evidence that extensive replacement involves, the determining causes of the form of the several ore bodies are obscure.

Structural Relations.

The main bodies lie in four groups, each marked by certain common features. (See Fig. 3.) The earliest ore mined came from the Southern Cross vein. This is a fissure cutting across the bedding, with a strike of N. 40° W. and a dip of 60° to the northeast. In the favorable magnesian limestones this fissure has been the nucleus of considerable replacement, but where it cuts across the shale member of the Hasmark formation it is a small, tight crack difficult to trace. The ore body of the Southern Cross vein has an average length in the limestone of 200 ft.

Probably associated with this in origin is the Pleiades vein, an irregular ore body replacing the limestone immediately above the Hasmark shale. It roughly parallels the bedding. This vein cannot be traced above the Southern Cross fissure, and is strongest immediately below it, passing into a narrow stringer along a bedding plane a short distance away. At its best it forms an ore body about 50 ft. in length.

The third series of shoots, collectively called the Glory vein, is likewise

parallel to the bedding. It outcrops 200 ft. west of the Southern Cross
vein, and, dipping to the west, diverges from it more and more with
depth. This ore is, like the Pleiades vein, near a shale-lime contact,
and follows, in a general way, the base of the lower Red Lion shale. The
chief ore shoot averages 200 ft. in length, with other smaller bodies along
the general zone.
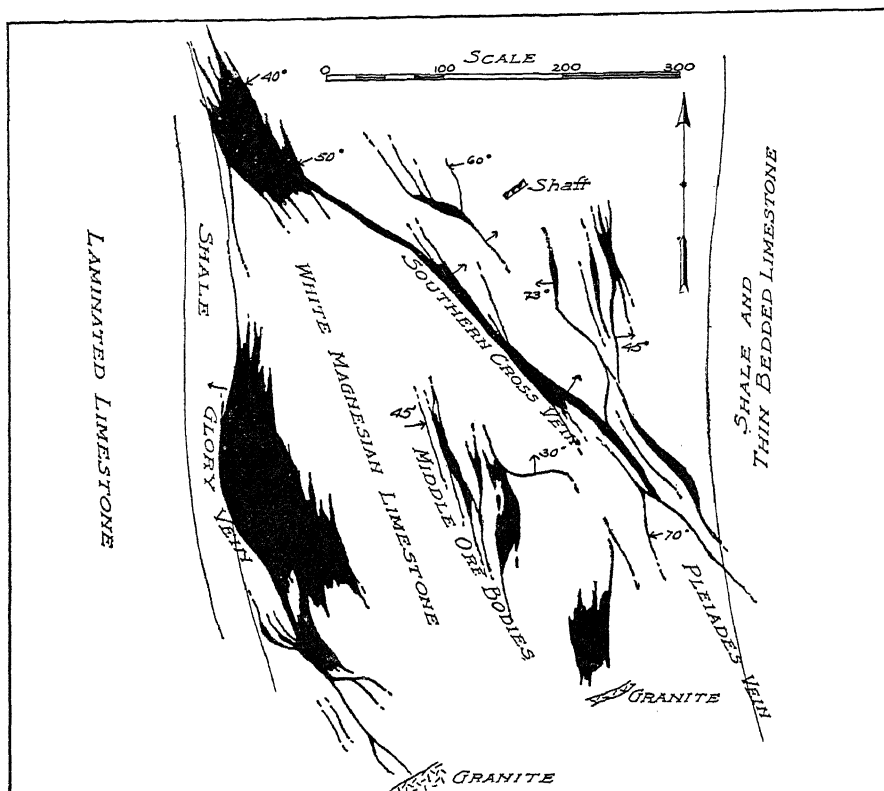
FIG. 3.—DISTRIBUTION OF ORE BODIES IN THE SOUTHERN CROSS MINE.

A certain broad system is thus discernible in the above groups of ore
bodies—a cross fissure dipping east, with associated replacement in
favorable strata; and replacement shoots running from this along lime-
shale contacts on both sides of the massive limestone member. The
fourth and most recently discovered group is far less simple in relation-
ship. It lies within the massive magnesian limestone itself, between the

Glory vein on the west and the Southern Cross-Pleiades ore bodies on the east. The ore bodies of this group show much greater replacement of the lime than any of the preceding, with less regularity of outline. The boundaries coincide locally with bedding planes, and locally with the small post-mineral fractures above mentioned, but most frequently cut across the bedding at irregular angles. These shoots lie in large, roughly lenticular masses pitching northward at flat angles. Their dimensions, as thus far developed, exceed those of the earlier ore bodies.

Origin of the Ores.

Any theory of the origin of these ore bodies confronts certain difficulties. The fact of limestone replacement is clear, but the source of the active solutions is less obvious. The coincidence in the entire district between the magnesian lime of the Hasmark, the edge of the granitic batholiths, and the iron ore bodies is striking. Many of the latter, however, are a thousand or more feet away from any exposure of igneous rock, so that if, as seems probable, the granite is the source of the mineralizing solutions, the replacement bodies represent a relatively remote phase of their action. At the actual contact there is a great development of calcite, wollastonite, garnet, and magnetite, and several large bodies of the latter extend well into the limestone. It is therefore possible that the lime-silicate zone, the magnetite bodies, and the replacement bodies of the Southern Cross represent progressive stages of contact metamorphism, proceeding from the contact itself out into the sedimentary rocks.

The general relation of the Southern Cross ore bodies points to the Southern Cross fissure as the most probable channel for the introduction of the mineralized waters. From this fissure the solutions worked out in such favoring localities as lime-shale contacts to form the Pleiades and Glory veins, while smaller fractures and bedding planes admitted them into the intervening limestone bed which now contains the middle ore bodies.

There can be but little doubt that the ore was originally deposited as sulphide, and that the present occurrence of the oxides is due to the action of surface waters. It is true that hematite is not rare as a mineral of contact metamorphism, and may have been present with the pyrite of these deposits, but the cores of sulphide found at such widely separated points as the Oro Fino, Southern Cross, and Red Lion mines can hardly be explained otherwise than as the remnants of the original ore. The oxides also frequently form pseudomorphs after pyrite. There is in addition much evidence that alteration from the surface is subsequent to the formation of the ore bodies. Natural caves in the limestone are of frequent occurrence, often in close proximity to the ore, and in no instance are the

iron oxides found filling such cavities. On the other hand, bodies of ore often form the hanging wall of the caves. It is probable, therefore, that these openings are largely due to the action of surface waters rendered acidic by the oxidation of the overlying pyritic ore bodies.

A few instances in which such caves have been filled with the débris of surface waters are so unlike the typical deposits of the mine as to be instructive in this connection. The best examples are found in the Oro Fino Red Lion mines, but they are not lacking in the Southern Cross itself. These caves are filled with a partly consolidated mass of heterogeneous boulders, cemented in places with iron oxide, but mostly imbedded in sand and gravel. The boulders comprise quartzite, granite, and shale, as well as limestone, and the source of supply is obviously the surface soil and wash. Many such deposits have been developed in the district, but none have contained commercial ore.

Summary.

The ore bodies thus far exploited in the Southern Cross mine, therefore, consist of the oxidized portions of pyritic replacement deposits in limestone. The ore was derived from the nearby granitic intrusions, forming the remotest, and probably the latest, phase of the contact metamorphism. Well defined cross-fissures have apparently admitted the solutions to the limestone strata most susceptible to their action, and on both borders of these favorable beds irregular deposits of gold-bearing pyrite replaced portions of the lime. Bedding planes and small cleavage cracks determined the extent of additional replacement within the limestone itself. With the geological evolution of the region the upper portions of the ore bodies became oxidized, and the resulting acidic surface waters formed caves throughout the limestone. Such of these caves as further erosion opened to the surface were filled with detritus.

The oxidation, and circulation of surface waters, has resulted in the leaching of the gold from portions of the deposits, and its concentration and reprecipitation in certain channels in the oxides. From these channels came the high-grade ore that periodically repaid the prospecting of the early operators, but under the conditions initiated by the construction of the railroad this search for enriched ore is no longer necessary, and in the genesis of the deposits, rather than in the nature of their enrichment, are found the clues to the successful development of the mine.

Timbering in the Butte Mines.

BY B. H. DUNSHEE, BUTTE, MONT.

THIS paper is not intended to be a technical discussion of square-set framing as used in mines, but merely a short description of the different kinds of framing that have been used in the Butte mines, and what has been decided upon, after years of practical experience, as being the best, taking into consideration the physical condition of the veins and country rock, and also the supply of timber available for mining purposes.

Butte is, essentially, a square-set district. Virginia City, Nev., was the first mining camp to use this method of timbering, and to Philip Deidesheimer, who went to the Ophir mine in 1860, is given the credit of solving the problem of square-set framing, where large bodies of ore were stoped and the timbers were supposed to hold the ground from caving without any filling. That he did his work well is shown by the fact that the main features of his framing are in use to-day.

In the Butte mines the conditions are such that this method of timbering is generally used. We do not, of course, depend upon the timbers any more than is necessary. The worked-out stopes are filled with waste rock as close behind the miners as possible and not interfere with mining operations. The waste rock is obtained from development work on the different levels or from the surface. In the earlier days of mining in Butte there were numerous independent companies and naturally there were different ideas as to the best way to frame the square sets, although the general features were the same.

Timber was abundant for mining purposes and the common practice was to use sawed lumber, generally 10 by 10 in. or 12 by 12 in. square. Some of the mines used for girts 6 by 10 in. or 8 by 12 in. timber, depending upon whether they used 10-in. square or 12-in. square timber for posts and caps. This avoided the necessity of framing the ends of the girts, and also required less timber for the

same work. It proved very satisfactory, as the cap, which is the stronger of the horizontal members of the square set, is placed perpendicular to the strike of the ore body and resists the pressure from the hanging wall of the vein; while the main function of the girt is to resist the side swinging of the caps as the weight from the walls comes on them.
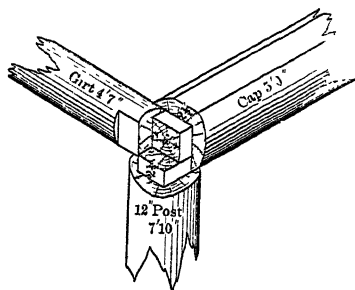
Fig. 1.—Stope Set.
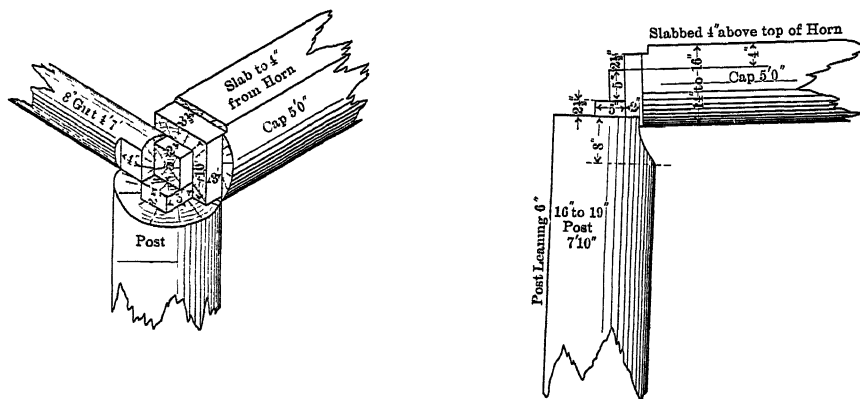
Fig. 2.—Sill Set.

FIGS. 1 AND 2.—TIMBER SETS AT GAGNON MINE.

The Gagnon mine was an exception in that round timbers were used for square sets long before it became a practice in the other mines. About 1886, while the management of the mine was under C. W. Goodale, a machine was installed to frame round timbers. With slight alterations, the same framing is in use to-day at this mine, as the Gagnon still has an independent framing mill. Fig. 1 gives a perspective of the kind of framing used at this mine at the present time. The post has a horn 5 in. square and $2\frac{1}{2}$ in. high on each end. The cap is similarly framed, but in addition has a shoulder taken off the bottom, 5 in. from the center, to allow the cap to fit snugly on top of the post, and on top, also 5 in. from the center,

a slab is taken off the full length to allow for a level floor in the stopes. The girt is generally less than 10 in. in diameter and is framed on two sides only. The posts are 7 ft. 10 in. in length, making the sets 8 ft. 3 in. from center to center. The caps are 5 ft. in length and butt end to end. The girts are 4 ft. 7 in. long. This makes the sets 5 ft. from center to center, either cap-way or girt-way. On the main working levels of the mine practically the same framing is used, with some slight changes which can readily be seen in Fig. 2. The timbers are especially selected and much larger, and the posts,
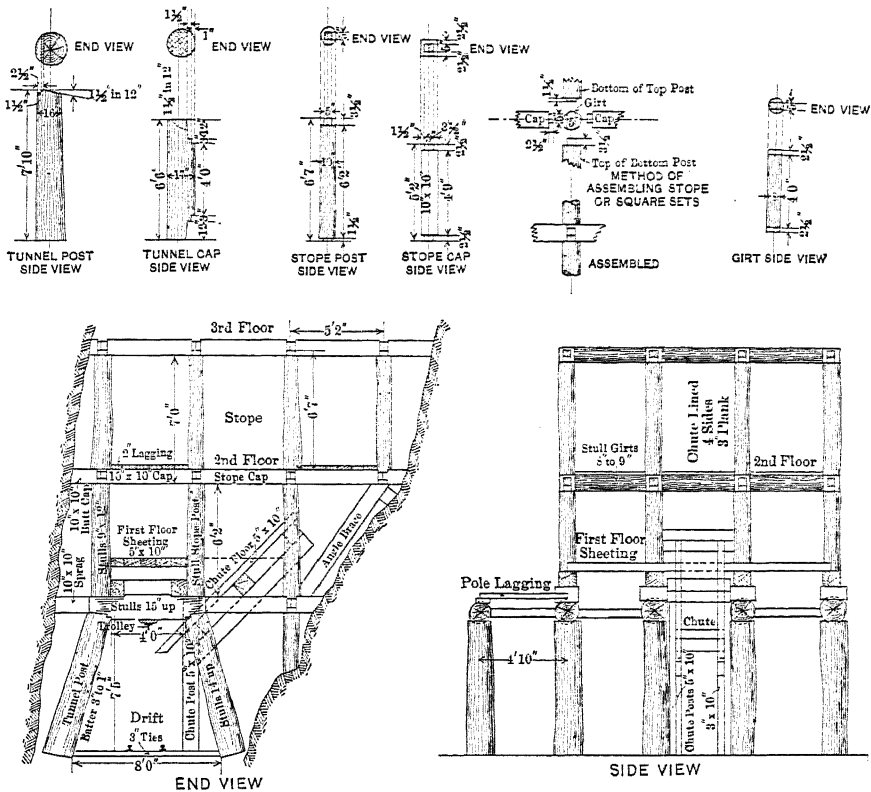
FIG. 3.—TIMBERING METHODS AT STEWARD MINE.

which are flat bottomed, are 1 ft. farther apart on the floor of the sill than they are at the cap. This gives additional strength to resist the side pressure which tends to push the posts into the drift, thus adding materially to the life of the drift before repairs are necessary.

The Steward mine also has an independent framing mill and continues the same method of framing that has been in use for a number of years. (See Fig. 3.) The posts have a horn 5 in. square by $3\frac{1}{2}$ in. at

the top, while the horn at the bottom of the post is 5 in. square by $1\frac{1}{2}$ in. It was supposed to be easier and quicker to stand a post in the stopes with a short horn than with a long one. It necessitates the horn of the cap being out of center, which does not weaken the cap, but, generally speaking, the more symmetrical the timbers, the easier they are to put in place in the stopes. The posts and girts are round timbers, while the caps are 10 by 10 in. square. The posts are 6 ft. 7 in. in length, making the sets 7 ft. from center to center. The caps are 5 ft. 2 in. and butt end to end. The girts are 4 ft. 5 in., making the sets 4 ft. 10 in. center to center girt-way. The timber for the support of the ground in the main working levels is especially selected from the best stulls, varying in size from 14 to 23 in. in diameter. The framing is simple and inexpensive. The posts are framed on the top end only, being sized back $2\frac{1}{2}$ in. and down $1\frac{1}{2}$ in. to form a shoulder for the cap. Back of this shoulder both the cap and post are cut on a batter of $1\frac{1}{2}$ in. to the foot, giving the post a pitch of 3 in. to 1 ft. toward the foot or hanging wall of the vein. This makes the distance between the posts on the floor of the drift 8 ft. While it might appear that the batter given the posts is too great, experience has shown such not to be the case in this mine.

The end view, Fig. 3, also shows the method of putting in the sheeting, above which the stope is filled with waste rock as soon as possible after the ore has been removed. This sheeting is placed at a sufficient height above the cap of the drift set to allow for repairs when necessary. The general practice is to use small stulls for sheeting, or larger ones sawed in half. There is also shown the position of short, flat-bottomed posts and angle braces; also the chute which is used for ore or waste, and is carried up with the stopes.

Figs. 4 to 8 show the different framing in use at several of the mines before the advent of round timber.

Fig. 4 shows the framing at the High Ore mine. The post has a long and rather small horn, which is very liable to break with heavy side pressure, and with heavy pressure from the top the horn tends to crush.

With the Syndicate group (see Fig. 5) we have, perhaps, the other extreme. The horn of the post is only 1 in. in height, giving a small shoulder for both the cap and girt; besides the framing of the girt is unnecessarily complicated, which would, of course, increase the cost of timbering.

Fig. 6 shows the framing at the Anaconda group. This is, I think, the best method of framing sawed timber for square sets. It is simple, cheap, and retains the full strength of the timbers in whatever direc-

tion the pressure may come.   The horns on the posts are 6 in. square by 2 in.   They are strong and give a good shoulder for the cap and girt.   The caps butt end to end with a horn 6 in. square by 3 in., while the girts are framed 6 by 10 in., with a 2-in. shoulder to fit. The girts are unnecessarily large compared with the other timbers in the set, and the framing, as shown in Figs. 7 and 8, which is the same as Fig. 6 except that the girts are 4 in. less one way and require no framing at all—is just as good and less expensive.

Fig. 7 shows the framing for 10-in. square timber.   The girt is 6 by 10 in., and 4 ft. 6 in. long.   The sets are 5 ft. from center to center, either cap-way or girt-way, and are 7 ft. 6 in. from center to center in height.
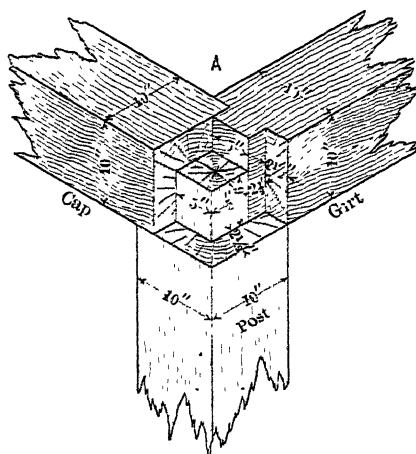
FIG. 4.—FRAMING USED AT HIGH ORE MINE BEFORE THE ADVENT OF
ROUND TIMBERS.

Fig. 8 shows framing for 12-in. square timber, where the girts are 8 by 12 in.   This size of timber was not used except in stopes where the ground was unusually heavy.

Timber suitable for the saw mill and available for the Butte mines became rapidly less, owing to the great and ever increasing demand from the mines.   There were extensive forests of pine and fir trees growing in the mountains surrounding Butte which were large enough for mine timbering if round timbers could be used for square sets; besides, the expense of timbering, which is a large item in mining costs, would be materially decreased.

The first machine that was installed in the Rocker framing mill, which is located at Rocker, about 3 miles from Butte, where most of the timber for the Butte mines is framed, was made to cut a strictly

mitered set.   The post had a flat top 8 in. square, with no horn, and
a miter cut to the outside of the timber.   The cap had a 5-in. square
horn, 4 in. on the top and bottom and $2\frac{1}{2}$ in. on the sides to the begin-
ning of the miter.   The girt had a horn which was $1\frac{1}{2}$ in. on top and
bottom, while on the sides the miter extended to the end of the girt.
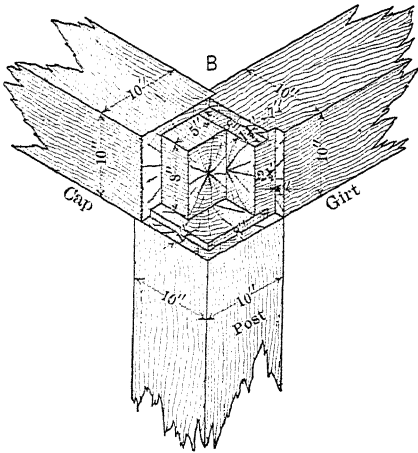
Fig. 5.   Fig. 6.
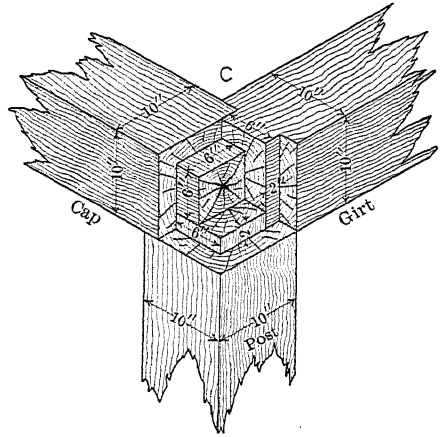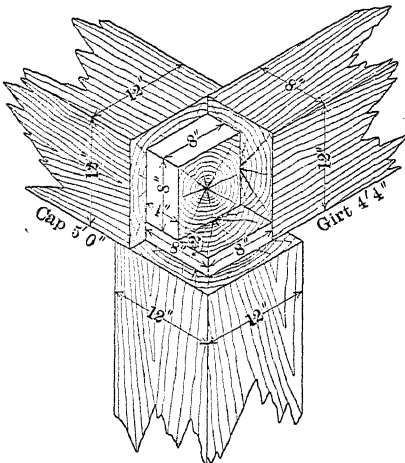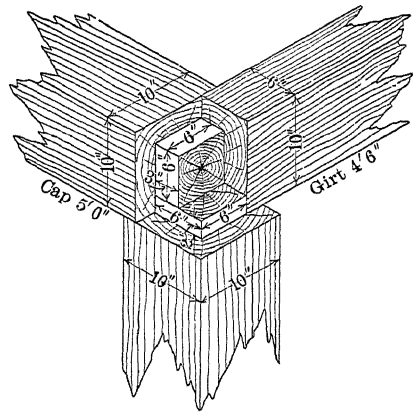
Post 7'0"   Post 7'0"
Fig. 8.   Fig. 7.

FIGS. 5 TO 8.—FRAMING METHODS BEFORE ADVENT OF ROUND TIMBERS.

This style of framing was used for some time in many of the mines,
but it was not easy to set the timbers in place and it was found diffi-
cult to block the timbers so that the sets would resist the pressure of
the ground satisfactorily.   Also the concussion caused by blasting
gave trouble.

An improvement was made in what was known as the combination square and bevel framing, shown in Fig. 10. The posts have a short horn 8 in. square by 1½ in., then the miter cut to the outside of the timber. The cap framing is somewhat complicated—the horn for a length of 1½ in. is 8 by 5 in., then for 2½ in. it is 5 in. square. At the base of the horn is a 1½-in. shoulder from which the miter begins.
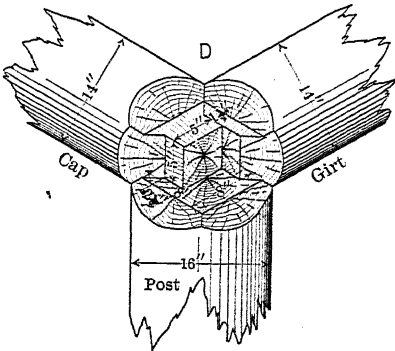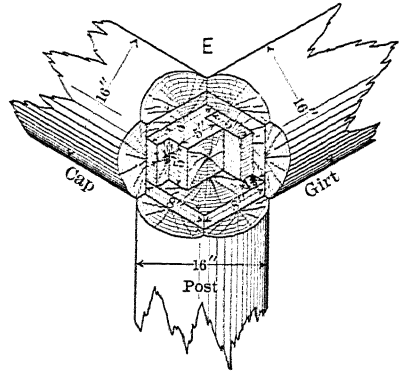
Fig. 9.   Fig. 10.

FIGS. 9 AND 10.—COMBINATIONS: SQUARE AND BEVEL, ROUND TIMBER.
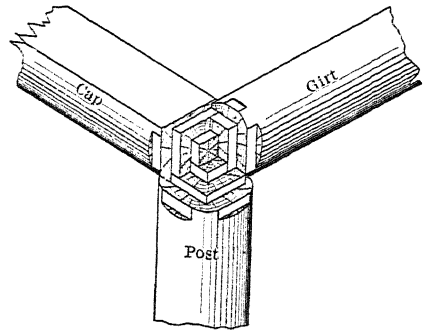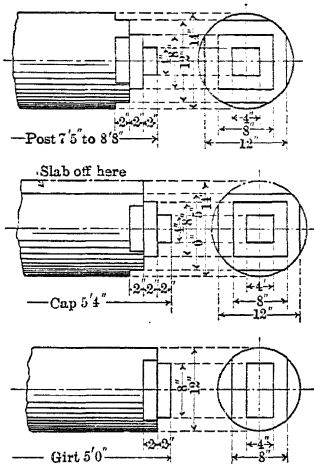
FIG. 11.—STEP-DOWN TIMBER FRAMING.

The girt has a horn 5 in. square by 1½ in., a shoulder of 1½ in., then it is mitered the same as the other members of the set. The posts are 7 ft. 5 in., making the sets 7 ft. 10 in. center to center. The caps are 5 ft. 10 in. and butt end to end. The girts are 5 ft. 5 in. These lengths make the sets 5 ft. 10 in. either cap-way or girt-way. These dimensions were the same in the mitered framing.

This framing, although an improvement, was not altogether satisfactory and a further improvement was made by eliminating the miter entirely, and having nothing but square shoulders, locally known as step-down timber framing. (See Fig. 11.) The post starts with a horn 4 in. square, then a step down of 2 in., and another horn 8 in. square, another step down, and, if the post is large enough, another horn 12 in. square. The cap is framed exactly like the post, and in addition one side is slabbed off 5 in. from the center of the stick to allow for a level floor for the stopes. The girt starts with a horn 4 by 8 in., then a step down and a horn 8 by 12 in., but the girt is the smallest member of the set and seldom reaches 10 in. in diameter. The size of the square set was changed as well as the style of the framing. The posts are 7 ft. 5 in., making the sets 7 ft. 9 in. from center to center. The caps are 5 ft. 4 in. and butt end to end. The girts are 5 ft. These lengths make the sets 5 ft. 4 in. from center to center either cap-way or girt-way. This change was made for two reasons : First, 5 ft. 4 in. square is large enough for most of our stopes, as the ground is frequently heavy; second, the common length for pole lagging, stulls and 2-in. plank, which is sometimes used for lagging as well as for flooring in the stopes, is 16 ft. and three pieces 5 ft. 4 in. long can be cut from a 16-ft. stick without waste.

This is, I think, the best framing for round timbers of various sizes. We obtain, as nearly as possible, the full strength of each member of the set regardless of the size of the stull. If the stulls were all of the same diameter we could have a more simple framing, somewhat similar to that shown in Figs. 6 and 7 for square timber. For the main working levels the timbers are especially selected, generally from 12 to 18 in. in diameter for posts and caps, the girts being smaller. The same framing is used, except the posts are framed on one end only. It is the intention ultimately to use this method of framing in all of the Anaconda Copper Mining Co.'s mines.

The framer used, shown in Figs. 12 and 13, is a new type made by Greenlee Bros. & Co., Rockford, Ill., from specifications and drawings prepared by the Anaconda Copper Mining Co. The first machine has been in operation only a few months, while the second one has just been installed. The framing is done by saws with insert teeth mounted to form a cutter head on a horizontal arbor. The round timbers are stepped down by a series of 2-in. shoulders. The cutter heads are made up to form a series of faces 2 in. wide with saws stepping down 4 in. in diameter each time. Each of these faces of the cutter head is made up of three saws equipped with insert teeth 0.75 in. wide, staggered 0.5 in. ahead of one another, and with the teeth over-

lapped so that they cut a face on the timbers just 2 in. wide.   There
are two arbors carrying the cutter heads at each end of the machine,
one above the other.   As the stull is fed through the machine it is
completely framed on the top and bottom sides.   The stick is turned
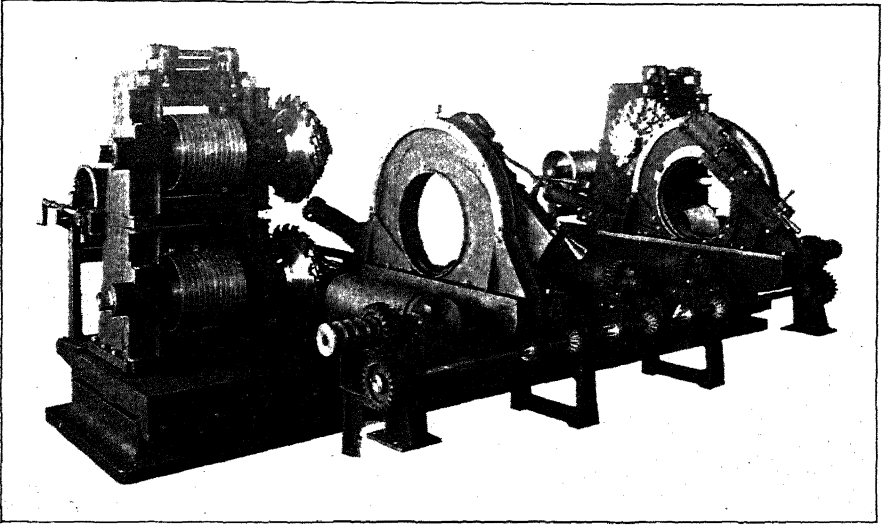
Fig. 12.
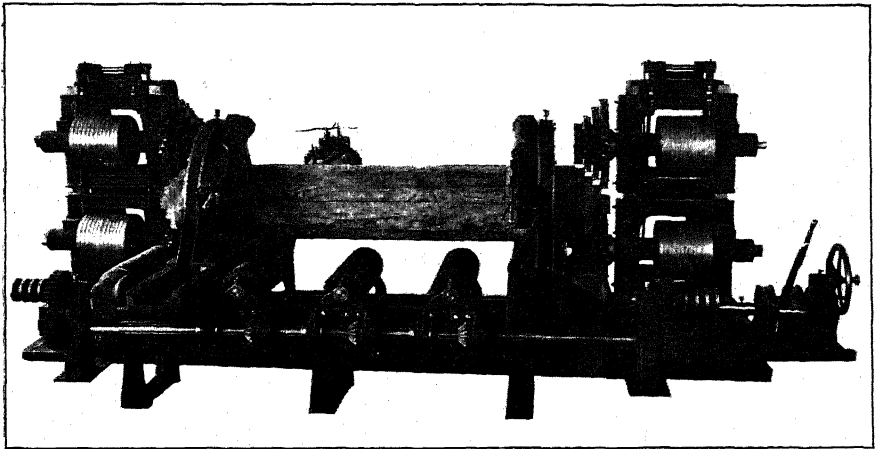
Fig. 13.

FIGS. 12 AND 13.—MACHINE FOR FRAMING MINE TIMBERS.

90° and again fed through the machine to complete the framing.
Over the saws are hoods connected with suction fans which take the
saw-dust and shavings to the fire boxes of the boilers.
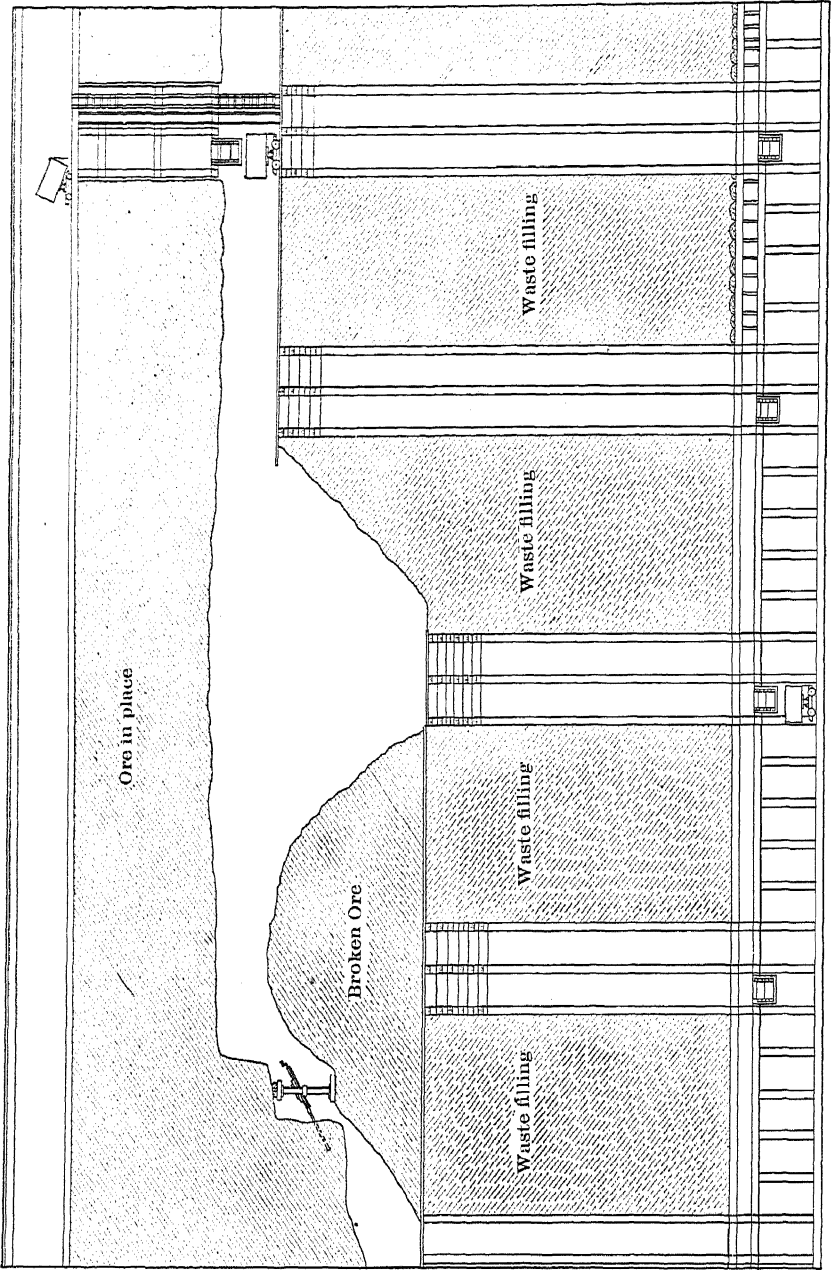
Fig. 14.—Back Filling System of Mining.

It might be interesting to describe a system of mining known locally as the "back-filling system," although it does not come under the head of timbering, but rather of mining without timbers. It is made use of whenever the physical conditions of the vein and wall rock are such that it is safe to do so. Both the ore and country rock must be hard and free from faulting planes. Fig. 14 gives a general view of this system of mining.

After a sill has been driven on the vein, a raise is put through to the level above. The ore on the first floor is then mined, and chutes started about every 25 ft. along the drift. Between the chutes, sheeting, composed of small stulls cut in half and resting on timbers a few feet above the drift sets, is placed from the foot to the hanging wall of the vein.

Starting from the raise the miners begin to break the ore. After the ore has been broken from foot to hanging for a short distance, the miners return, set up their machine on the broken ore, and again work their way forward. This is done two or three times until the broken ore is 15 or 20 ft. high, then the miners start again farther ahead and the shovelers begin to shovel the broken ore, first into the raise chute, until they reach the next ore chute. As soon as they have passed the ore chute, a temporary opening is made in the raise chute and waste is dumped into the chute from the level above and allowed to run on to the sheeting as long as it will; then a track is started about 6 ft. from the top of the stope, the waste is taken from the chute in a car and the filling of the stope continued to the first ore chute. The shovelers being now out of the way, the timbermen build up the ore chute one compartment wide and two in length, using small round stulls, 6 to 8 in. in diameter, to the level of the waste track. The waste-men then continue filling the stope, following the shovelers.

When the miners have gone as far as the stope is to be worked, they return and start another floor on top of the waste filling, which has been covered with a floor of 2-in. plank to keep the ore clean.

If at any time the ground changes and gets heavy, square setting can be started on the waste filling, and this is generally done when the stope has been worked out to within a floor or two of the level above.

For particulars regarding square-set timbering reference is here made to a series of articles by Claude T. Rice.[1] These articles were largely made up from data furnished by the author of this paper.

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[1] *Mining and Engineering World*, vol. xxxvii, pp. 1079, 1133, 1175; vol. xxxviii, pp. 3, 243, 295, 379, and 429 (Dec. 14, 1912, to Mar. 1, 1913).

DISCUSSION.

GEORGE E. MOULTHROP, Butte, Mont.:—The recording for the first time by Mr. Dunshee, in a readily usable form, of the various systems of square-set framing applied in the Butte district is a valuable contribution to mining information. The many different methods of framing employed, due to the individual ideas of superintendents and foremen of segregated companies, have given full opportunity to determine the best framing for the conditions as found in these mines, and the final adoption of a system uniform for all the mines has come about through a study of these various methods and an experience covering a period of many years.

The sawed 10 by 10 in. and 12 by 12 in. timbers undoubtedly give the best satisfaction; and the style of framing the sawed timber that under all conditions has given the best practical results is that used in the Anaconda group and other mines, described and recommended by Mr. Dunshee. In this framing the posts have a horn 2 in. high and 6 in. square, the cap end having dimensions of 3 by 6 in. Instead of the framed girt shown in the drawings of the Anaconda group, the 6 by 10 in. girt adopted at some other mines, which fits into the cap and post framing with no other preparation than cutting the timber to the necessary length, fills all the necessary requirements for strength and affects a material saving in timber and labor. This, as Mr. Dunshee states, is without doubt the type of framing which would be adopted as a standard for 10 by 10 in. sawed timbers if sawed timber sets were to be used in the future generally. But sawtimber for the mines was becoming scarce and expensive, and round timber for the square sets furnished an alternative which provided, at greatly reduced cost, material which in every way supplied the requirements of the mining conditions.

The use of the round timber with a satisfactory framing makes available a conveniently located and large source of supply which up to this time has served no useful industrial purpose. Particularly is this true of small timber, which is still standing in great quantities in the forests where the trees more desirable for commercial purposes have already been cut and marketed.

Then, too, timber which is damaged by fire and down-timber which is in a fair state of preservation can be framed for the mine sets and used to as good advantage as the best fir timber in stopes which are filled with waste material almost as soon as the timbers are put in place by the miners.

Of the three forms of framing round timber which have been tried and used extensively in the mines, the present so-called step-down

system is the only one which, after a year's trial, appears to give as much satisfaction as any framing used on 10 by 10 in. sawed timber. Mine foremen and miners use the sets framed after this pattern without opposition and the mines generally are visited with fewer caves than when the highest grade of sawed fir timber was plentiful and used exclusively.

The framing first adopted, in the so-called mitered sets with beveled surfaces, prior to the adoption of the square-cornered, step-down form of framing, failed completely in its practical application in the mines. This was due to the impossibility of blocking the sets rigidly enough to withstand the shock of ordinary blasting, and it took considerably more time to stand and block them even insecurely into position.

As the wall rock of many of the principal veins is never solid material, but to a greater or less degree mineralized granite which on exposure to air disintegrates and falls down from the walls, the most careful blocking of the sets of any style of framing is found in a short time to have dropped from place, the ground having fallen from behind the blocking. A slight weight of ground settling on the timbers had the effect of moving laterally the beveled posts and to such a degree that a cap or girt would fall out of its position, and when this happened, even if in only one set of a large stope of 50 or more sets, the concussion of a blast caused all the sets to fall to the floor of the stope. On this account, following the regular timbering gang, it was necessary to employ extra crews whose duty it was to examine and re-block the sets. On the other hand, with the square-corner framing the experience has been that with the falling out of the blocking of even several sets of a stope the weight of the ground takes effect more nearly vertically, and instead of the posts easily sliding laterally on the beveled surfaces, the weight binds the sets tightly together, and no amount of blasting ever sweeps down any great number of sets of a large stope, as was the common and usual case with the mitered sets.

More care and time are necessary in standing a set of round timbers than with the sawed timber sets, as with the latter it is an easy matter to stand a post in exact line with the sets already in place by sighting with the eye along the faces of the standing and blocked sets. With the round timbers, which are always of varying sizes, this method is impossible, and the miner, after temporarily placing the several members of the set into their approximately correct positions, proceeds to block the set and by wedges brings the framed ends into a close and even fitting. The fitting of these unions is the indication

of the accuracy of the square formed by the timbers. The measurement of the diagonals is a common and the best way of determining the correctness of the position of the set.

With the drift and sill-floor posts, which are flat-bottom, being framed at the top end only, the common practice is to first place as a sill a 2-in. plank, on the ends of which rest the posts. The position of the plank is determined by a carpenter's level or a grade-staff to obtain the correct elevation for the desired grade—0.25 to 0.5 in. per set—the alignment being determined by bringing the center of the plank to a point in line with two plumb lines hung from the centers of caps in sets already standing in the same row. Where ore chutes are to be built in, in the sets, and these are often as frequent as every 25 ft., 10 by 10 in. square timbers are used, instead of the round timbers, in order that a smooth surface may be had to which is spiked the 2 or 3 in. plank lining. These 10 by 10 in. square sets for the ore chutes are carried from one level to another as the stope progresses, so that every fifth or sixth set, instead of being of round timber, is of 10 by 10 in. timber framed like the round.

Round timber is sometimes hewed so that the chute lining may be spiked to it, but in heavy, moving ground the chute in square timber stands better and lasts longer.

Shaft-Sinking Methods of Butte.

BY NORMAN B. BRALY, BUTTE, MONT.

THE following is not offered as an extended paper on the subject of shaft sinking, but more as a description of the present practice of shaft sinking in the Butte district.

The Anaconda company is sinking at present the following shafts: Badger State, Moose, Mountain Consolidated, Mountain View, Pennsylvania and Tropic. At the High Ore and Leonard mines the shafts are being deepened by sinking two compartments and will subsequently be enlarged to the full working size by raising alongside the compartments which are now being sunk.

The following other companies are also sinking shafts: Butte-Alex Scott, Butte and Zenith City, Boston and Corbin, North Butte Mining Co. (at the Granite Mountain shaft), East Butte Co. (at the Pittsmont shaft), and the Rainbow Lode Development Co.

Shaft Timbering.

The shafts of the Butte district, generally speaking, are of three compartments arranged in a straight line, and the pump compartment is usually arranged for the use of an auxiliary hoisting cage, as well as for carrying pipe lines and electric wires. The Rainbow shaft is practically square, and of three compartments. The length of the shafts outside of timber will vary from 17 to 22 ft., and the width from 6 ft. 6 in. to 7 ft.

Fir timber is used in all shaft work. This may be obtained from the Coast, when it is known as Coast fir, or from Idaho or Montana, when it is known as native fir. The native fir seems to be the tougher wood, but it is hard to obtain clear lumber in the lengths and sizes necessary for shaft work. Some of the shaft timber of the Anaconda company has been treated with creosote as a preservative, but this practice has not come into general use, as the ground of the camp is so heavy that the timbers usually crush before they rot. The size of timber most used for shaft work is 12 by 12 in., while 10 by 12 in., 10 by 14 in. and 12 by 14 in. are used for some of the special pieces around the shaft. The new Leonard shaft is timbered with 14 by 14 in. wall plates throughout.

No machinery has been introduced here as yet for the framing of shaft timbers, although the North Butte company is considering such a machine, and all of the timbers are therefore hand-framed. It may be of interest to note that $\frac{1}{2}$ to $\frac{1}{4}$ in. is generally held off the wall plate and end plates at each end in the framed part to make the mitre draw close together when the set is being blocked; thus the wall plate for a shaft 20 ft. long would be 19 ft. 11 in. long. In order to hold the set together when it is being blocked, 1-in. wooden pins are driven at each corner, in holes bored for that purpose. These pins are made by driving 1-in. square blocks of wood through a 1-in. round hole in iron 1 in. thick.

To keep the shafts vertical, blocks nailed together (Plate I, Fig. 2), made of boards $\frac{1}{16}$ in. thick are used, so that half the width of the line will bring the center of the line 1 in. from the corner of the set both ways. While the timber is being framed, the carpenter marks with his scratch awl a line on the timber 1 in. from each corner, as shown in Fig. 1, Plate I, and this line saves a great many rule measurements in the early blocking of the set. At about every 50 ft. these lines must be lowered to new blocks. A set is selected at a point about 50 ft. below the blocks, and measurements are taken each way from the corner line to determine how far the line is from the timbers. The new blocks are then nailed in place and the lines suspended from them. If the measurements to the timbers do not correspond to the old measurements, the block is cut with a pocket knife to bring the lines exactly to their former position. It is well to measure diagonally across between lines at about every 100 ft. to see if the shaft remains square.

A transit should be taken down before reaching a station, and readings taken from the corners down to the level of the instrument. Posts are then cut to bring the shaft timbers level. This is called a leveling-up set; this leveling may be done with a carpenter's level, but not as quickly nor as accurately.

The placing of the collar set is the first thing to be done in starting a shaft. Its height above the ground is determined by the amount of room available for the dumping of waste rock from the sinking operations. In a level place it is generally raised about 20 ft. The wall plates of the collar set are usually made 10 ft. longer than the shaft set, projecting 5 ft. at each end. This set bears the weight of the shaft sets below, and usually of the gallows frame. Posts resting on blocks are placed under each end, and diagonal braces are then placed around the shaft to carry part of the weight and prevent the set from moving. Figs. 3 and 4, Plate I, show the collar set of a two-compartment shaft, with the diagonal braces and posts.

The size of the guides varies greatly. At first 4 by 6 in. guides were used almost universally in the camp, but lately the size has been
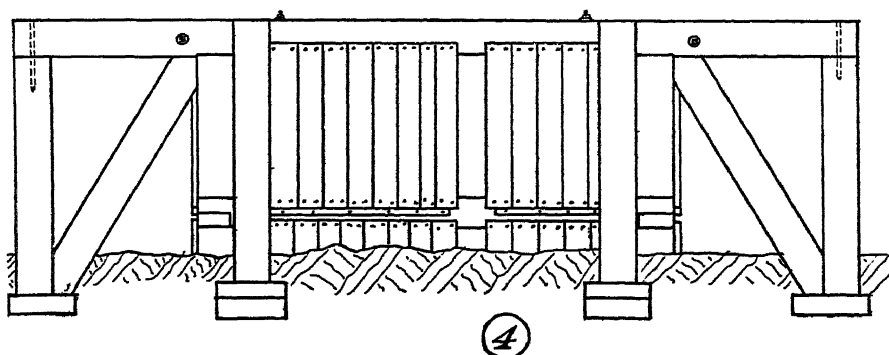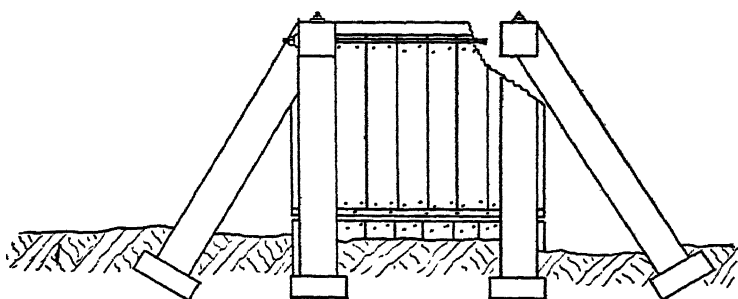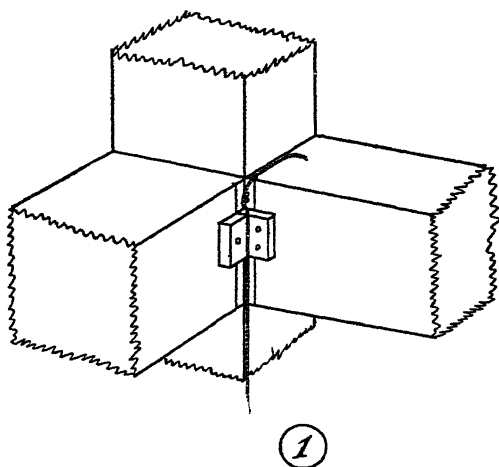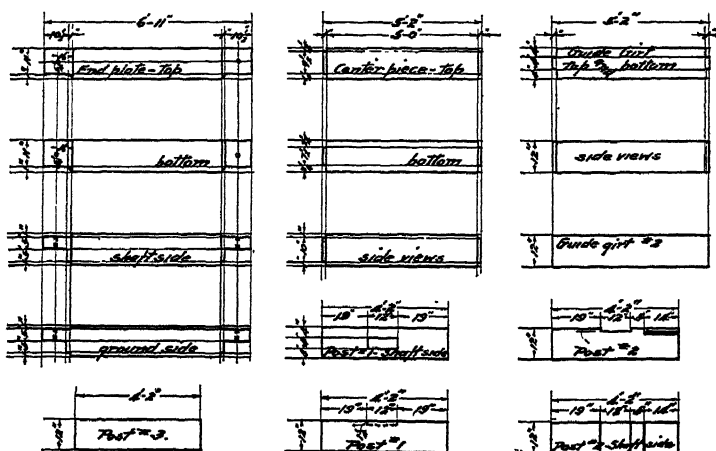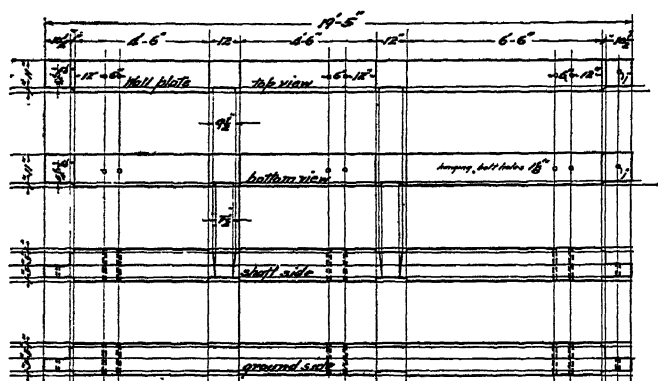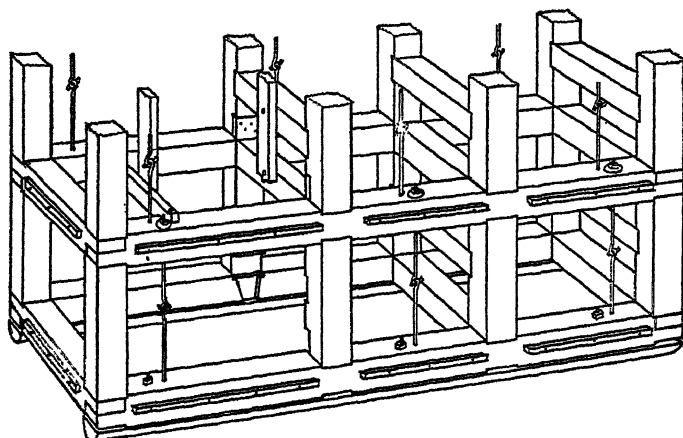
PLATE II.—GRANITE MOUNTAIN SHAFT OF THE NORTH BUTTE MINING CO., USING GUIDE GIRTS IN THE HOISTING COMPARTMENTS. 12 BY 12 IN. USED THROUGHOUT.
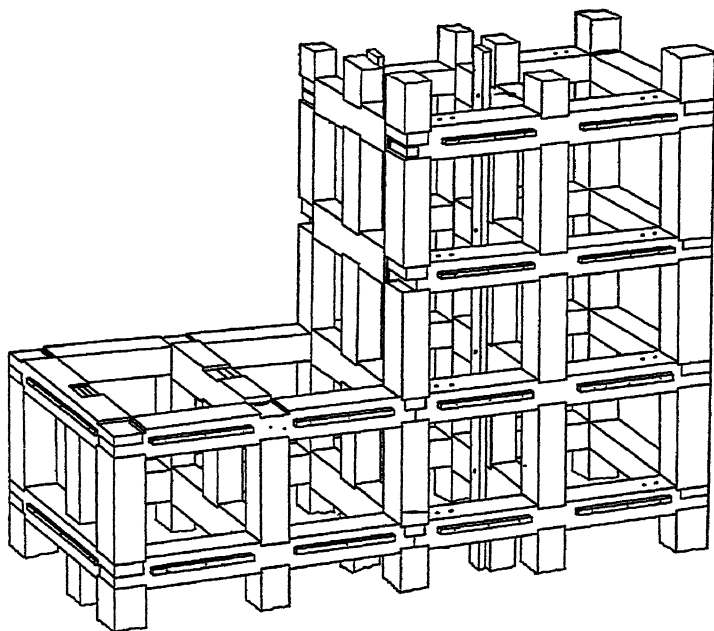
increased to 4 by 9 and 5 by 9, held by ⅞ by 8 or ⅞ by 9 in. lag screws. The larger size guide can stand heavier loads and greater shocks, and two lag screws are used in each timber; the holes, two in number, are placed diagonally across the guide, and, after a number of years, they are placed diagonally across in the other direction, giving fresh timber for the screws to hold to.

Guide posts are vertical posts running behind the guides to give them more support. They may be seen in Plate III. They are framed into the centers and end plate of the shaft. Where used they are carried in the main hoisting compartments, and sometimes in the auxiliary hoisting compartment, but not in the pump compartment. They are the same width as the center, and usually 12 in. in the other direction.

Guide girts are girts running horizontally between posts and equidistant between the centers of the shaft. They are framed into the posts of the shaft, as seen in Plate II, giving support to the guide and also helping to take the side weight of the shaft. They are being used at the Granite Mountain shaft of the North Butte company, and will be used in the retimbering of the Neversweat shaft of the Anaconda company. In heavy ground the guide girt would seem to be more satisfactory than the guide post.

To allow for the end plate of the shaft pushing in, 2 in. is often allowed in the outside hoisting compartment. To make the distance between guides the same size in the two compartments, a 2-in. block of wood is placed between the guide and the end plate of the shaft. As the end plate comes in, the block is cut down or removed until the guides are again the right distance apart. This is called a filler. Sometimes, as at the Badger State mine, the filler may be a vertical post carried up the shaft against the end plates, and bolted with ⅞-in. bolts to the guide posts. At this mine a 10 by 12 in. post, in 20-ft. lengths, is carried up alongside the guide posts, and a 6 by 12 in. post is carried up in the other hoisting compartment on the opposite side. At the Moose shaft these fillers are carried behind each guide, the same size for both sides of the hoisting compartment, and are, I believe, 4 by 12 in.

In sinking a shaft spliced wall plates are often used. In heavy ground the ground cannot be kept open to a depth to allow a full length wall plate to be turned in a shaft so as to be brought into place and, as long length timber is hard to obtain, the splice is coming into more general use. Four hanging bolts must be used on a side with spliced wall plates. The splice, shown in Plate V, Fig. 1, is the one which finds most general use. In this the center is supported by the wall plates while the set is being blocked. Fig. 2 shows the splice when one side of the shaft is to be raised. It will be noted that the mitre is left out and the framing is straight. This is because dirt getting into the straight joint will not throw it out of line
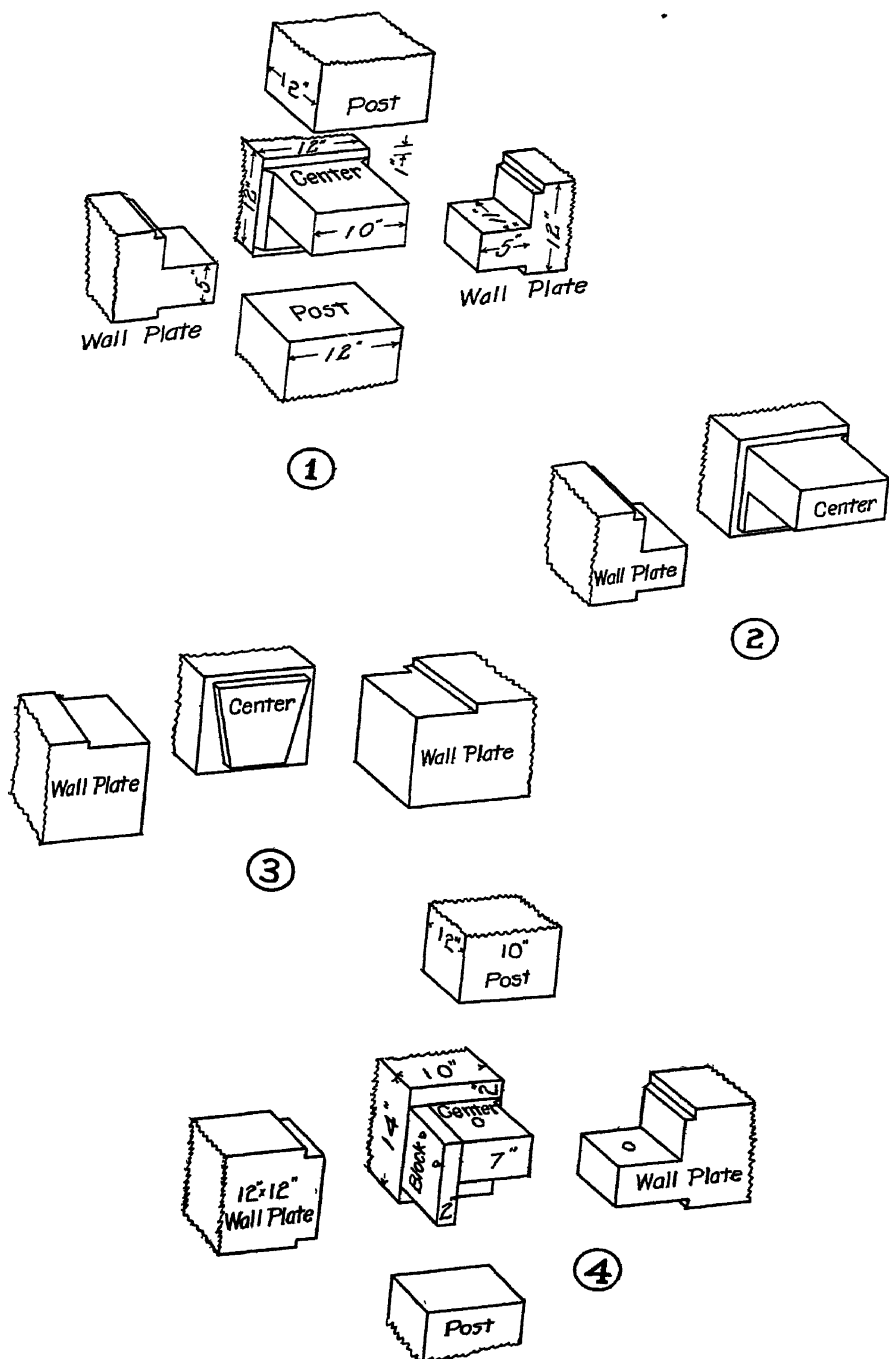
PLATE V.

1. Usual method of splicing wall plates.
2. Splice where half of shaft is raised.
3. A common method of splicing wall plate.
4. Method of splicing wall plate at High Ore mine.

as much as it would the mitred joint. When the shaft is being sunk a
block of wood is placed in the opening under the center, and, when the
other half of the shaft is raised, this block is chopped or picked out and
the other half of the wall plate placed in position.

Fig. 3 shows a spliced joint very easy to frame. The center in this case,
however, has to be supported while the set is being blocked. This is
usually done by nailing it to the wall plate before the posts have been
placed in position. At the Tropic shaft the set is blocked on both sides
of this center temporarily; the end blocks are then set, and the blocks
on each side of this center are then taken out and the set brought to line
and blocked permanently. A temporary post is sometimes placed under
the center to hold it up while being blocked. This method of splicing
is cheaper to frame, but a little more expensive to put in place.

Fig. 4 shows the method of splicing at the High Ore mine. A 2-in.
block is nailed to the center before it is sent down into the mine and the
corners of this set are held together with pins. The other half of the wall
plate can be easily pushed in, as the distance is small. This is not as
strong a splice as the one shown in Fig. 1, but would be excellent where
a small shaft is being sunk and there is a possibility that more compart-
ments will be added. On the whole, it may be said that it is cheaper
to sink the whole shaft than it is to sink part of it and raise the balance.
In soft ground where it is not desired to open up too much country at
once, the Anaconda company finds it an advantage to sink and raise.

The timbering takes place from the top of the muck pile whenever
possible. That is, the dirt, after blasting, is mucked down to about
10 ft. below the last set. The new set is hung and blocked; then mucking
is continued. The men do not have to build any staging to drive the
wedges, which are driven from below. If the timber is kept close to the
bottom it is sometimes found after blasting that the muck is pushed
up to within a few feet of the last set of timbering. If the men could
work only in one compartment mucking would be very slow; so, to avoid
such a possibility and to aid in turning the timbers to get them in place,
one of the centers of the shaft on either side of the hoisting compartment
is left out on the bottom set. A 2-in. strip is cut from the post and the
center is pushed in through this opening and hammered down into place
when the set below is put in. A block made to fit the opening in the
post is then nailed into place, completing the set. This may be seen in
the Rainbow shaft set, Plate IV, and the Granite Mountain set, Plate II.
In the Granite Mountain shaft it will be noticed that the guide girt is
also left out. The guide girt is slipped in from the side; then it is wedged
on the end and the side weight of the shaft soon makes it solid. The
center and girt are always kept within one set of the bottom timber;
this style of center is framed with a dovetail.

Wall Plate, - Top View

Wall Plate - Bottom View

Wall Plate, - Shaft Side

Wall Plate - Ground Side

End Plate, - Top View

End Plate, Bottom View

End Plate - Shaft Side

End Plate Ground Side

Spreader - Top

Spreader - Bottom

Center Piece, Top View

Center Piece, Bottom View

Center Piece, Chippie Comp

Center Piece, Hoist Comp

End Plate, Top View

End Plate, Bottom View

End Plate, Shaft Side

End Plate, Ground Side

Spreader - Side

12"X14" Post

12"X14" Post

12"X12" Post

12"X12" Post

PLATE IV.—RAINBOW SHAFT. AN ALMOST SQUARE SHAFT OF THREE COMPARTMENTS.
12 BY 12 IN. AND 12 BY 14 IN. WALL PLATES.
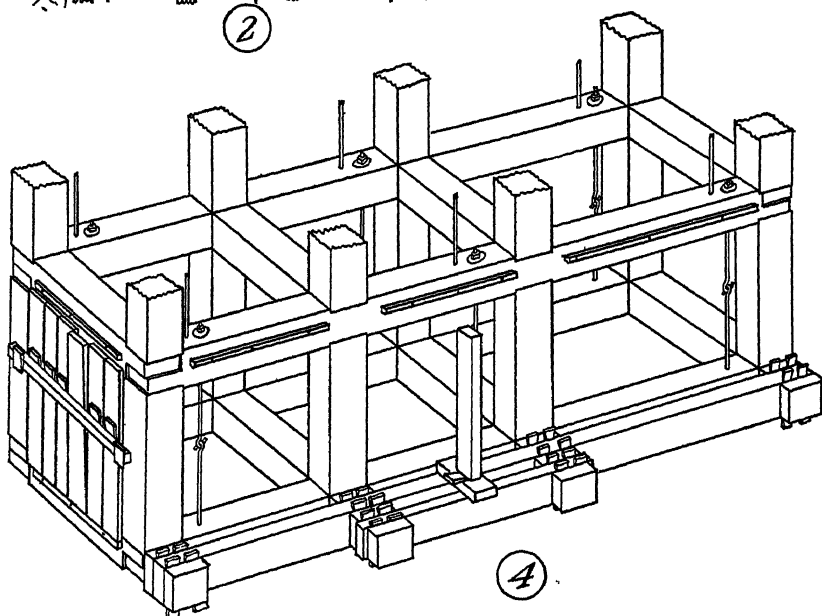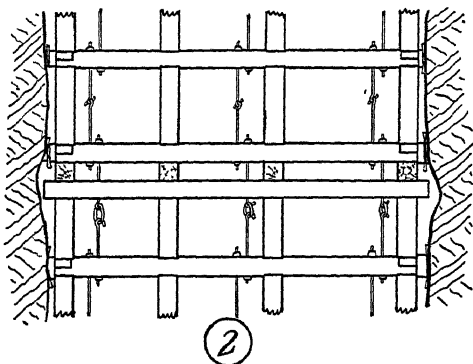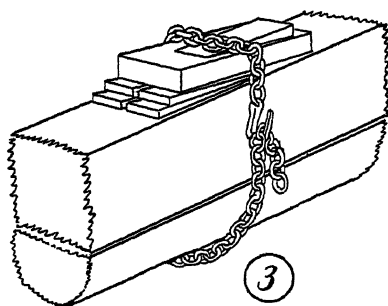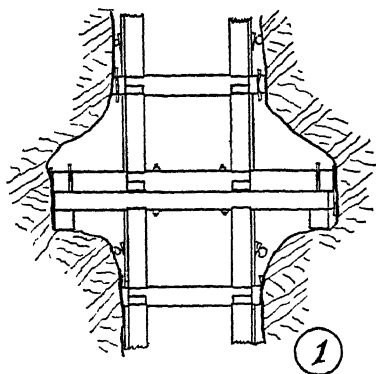
PLATE VI.

1. Shaft bearers, end view.
2. Shaft bearers, side view.
3. Method of fastening blasting logs with chains.
4. Method of blocking and lagging set.

Where the guide post is framed into the center with a mortise and tenon, the center cannot be slipped in from the top. It is then framed with straight sides and is pushed up from the bottom and held in place by the posts of the set below when they are being placed. This may be seen at the Mountain View mine. This method of pushing the center up from below is sometimes used at other shafts on account of the cheaper framing. The first method is considered the best for holding out the center. It sometimes happens that the wall plates of the shaft have been
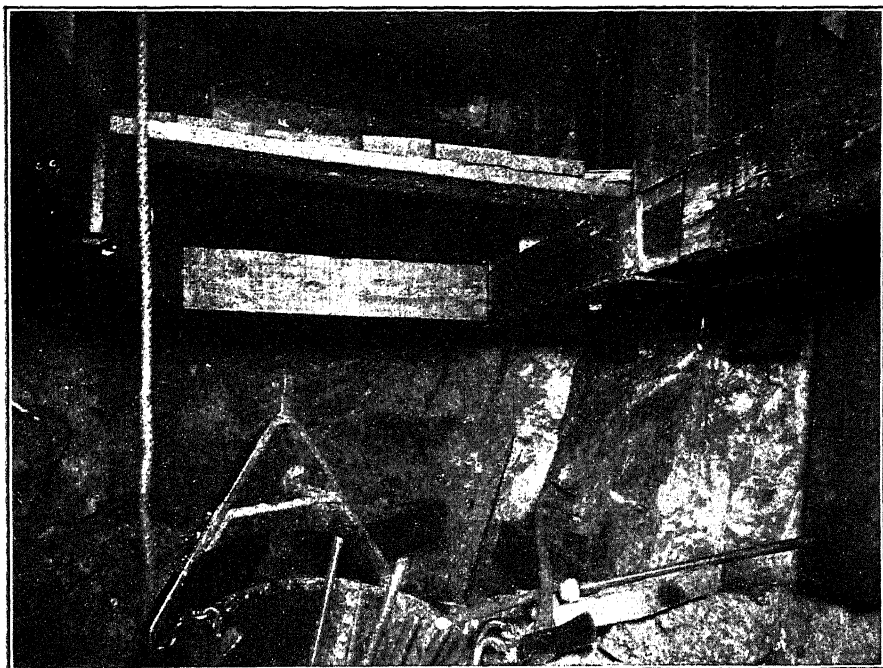
PLATE X.—TAKEN WHILE SET WAS BEING BLOCKED. GRANITE MOUNTAIN SHAFT, NORTH BUTTE MINING CO.

pushed in so that the center is hard to put in place. A small bar of 3-in. pipe, similar to a stoping bar, is then used to spread the timber.

The method of blocking and lagging the set may be seen in Plate VI. Where the distance is small,—say, not over 1 ft.,—a single block is placed between the set and the wall, the wedging being done between the block and the wall. When, however, the ground breaks back farther, a block of about 5 in. thickness is placed back of the center and toe-nailed from above. Blocks of the same size are placed along the side and toe-nailed. A stringer the length of the shaft is then placed along the side and nailed to the blocks. Blocks are again placed, and then stringers as needed. The wedging is then done between the blocks and stringers. The wedges

should be driven from below as well as above. An upright post is then put in place and wedged to the blocking above to prevent the blocking below from being knocked out by the blast. After the set has been hung and the posts are all in place, the set is generally raised up by placing a pinch bar under the hook on the hanging bolt; then the set is raised and the nut run down as far as possible by hand. In case blasting chains are used the bar may be supported by hooking the chains around
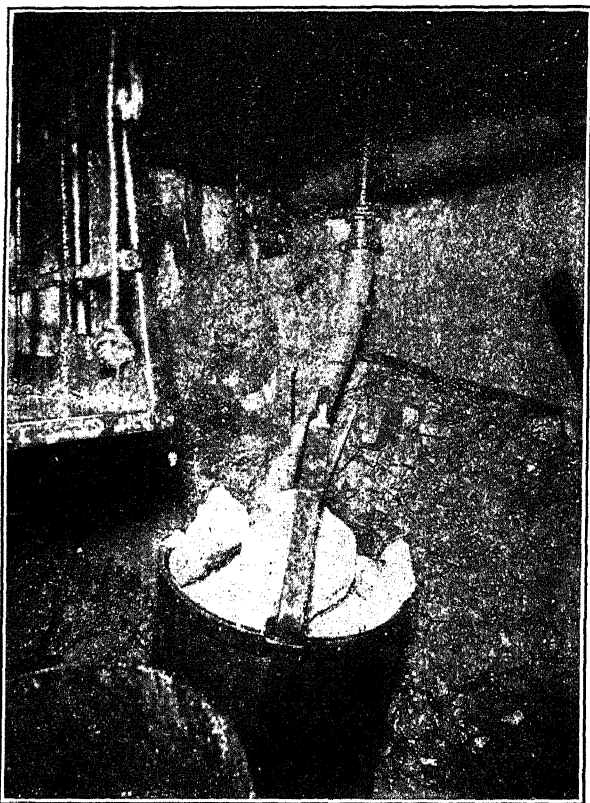
PLATE XI.—BOTTOM OF RAINBOW SHAFT, WHILE MUCKING.   BLASTING LOGS HELD
WITH CHAINS.

the hanging bolt above. In case bolts are used a U-shaped bar of $\frac{3}{4}$-in. iron, with hooks on the side, is made for this purpose. The top end of such a bar may be seen in Plate X, on the right-hand side of the bucket.

Two-inch fir is used for lagging. A strip is placed horizontally, back of where the lagging will come and about half-way between wall plates. The shaftman uses his ingenuity to find some method of support for this stick. The lagging timbers are then placed in and wedged from the back, the shaftman leaving one plank out for this purpose. This last

lagging is wedged from the inside of the shaft. Two-inch square strips support the lagging and are nailed to the wall plates before they are sent into the mine.

Blasting logs protect the timber from being cut up by the blasting. They are fastened to the bottom set with bolts or chains, as shown in Plate XI, which was taken in the bottom of the Rainbow shaft. Timber 8 by 12 in. is often used for this purpose; or a 14-in. stull, split lengthwise, makes an excellent blasting piece, as the round surface tends to throw off the rock. One-quarter inch iron and 3 by 3 in. angle iron, placed along the edge of the shaft timber, are also used. The iron blasting pieces, however, are twisted out of shape by heavy blasting and must be sent to the surface frequently to be straightened by the blacksmith. Where bolts are used to hold blasting pieces, they are of 1-in. size, and the head of the bolt is always placed down. Plate VI, Fig. 3, shows the method of tightening a blasting log held with a chain. The blasting log is held up, the chain is hooked over the timber, and a block is then slipped under the chain. Wedges are then driven under the block; a few blows with an ax makes them tight, and a side blow loosens them again when necessary. This is probably the best and most reliable method of holding blasting logs.

Figs. 1 and 2, Plate VI, show what is known as a shaft bearer. About every 400 or 500 ft. the shaft is widened; sills are laid along the side of the shaft as shown, and cross pieces are then bolted under the end plates and centers of the shaft, resting on the sills. The shaft set is then blocked and wedged on top of these cross pieces. Regular posts are then placed under the bearers when sinking is resumed. Shaft bearers have not come into general use, although they may be seen at the Leonard and Badger State shafts of the Anaconda company. The object of the bearer is to prevent the shaft from sinking on one side and to give additional support to the shaft below when sinking.

I will not enter into a description of any particular shaft, as it is hoped that the drawings will make themselves clear.

Sinking Equipment and Methods.

The No. 3 Rand, E 44 Sergeant, and jackhamer drills are the ones most in use in shaft sinking. With the reciprocating drill the tendency is toward long rounds, 10-ft. rounds being broken where the rock is hard; with the jackhamer drill a 5-ft. round is drilled. All sinking is done with cross bars where the reciprocating drills are used, and these bars are generally made 6 in. longer than the width of the shaft timber, while a longer bar and a shorter bar are used for special set-ups. The bars are made of extra heavy 4-in. pipe.

PLATE XII.
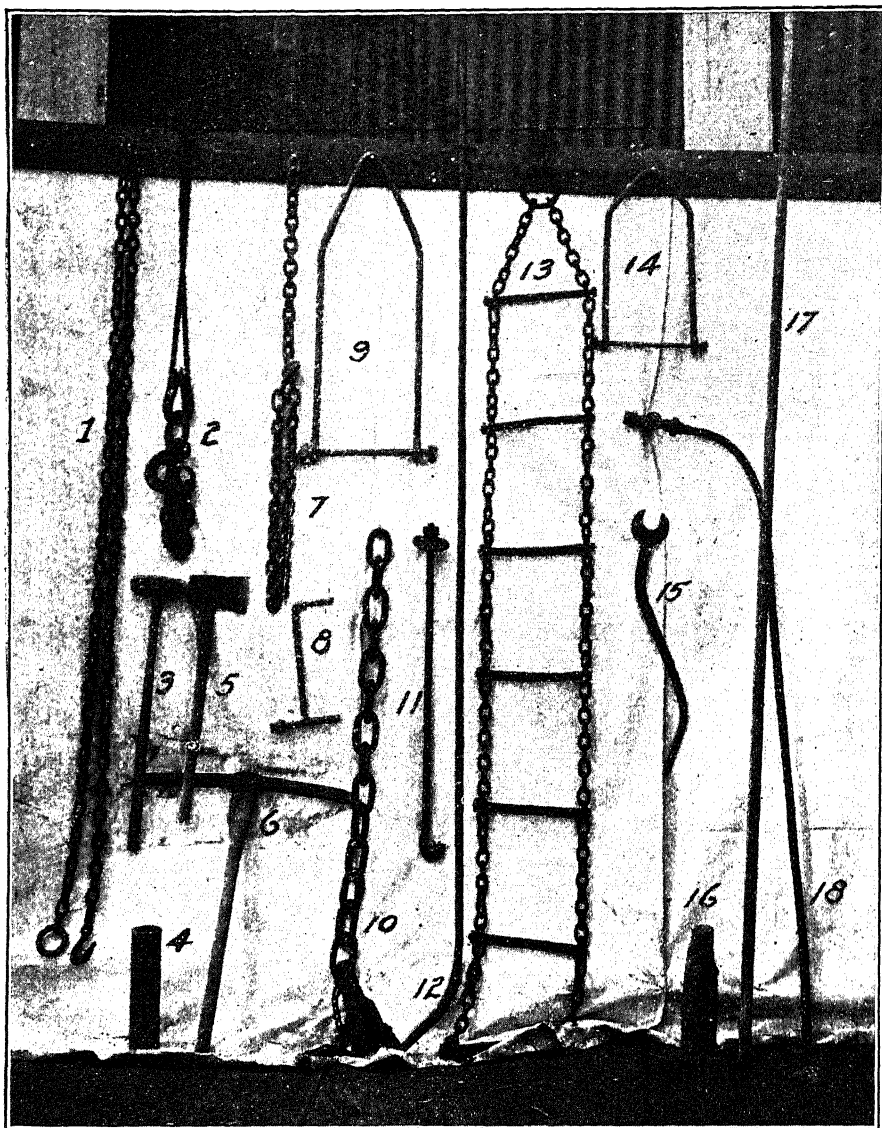
1. Rock Chain.
2. Flat Link Method of Fastening Bucket.
3. Rock Hammer.
4. Pipe cut for Collar of Drill Holes to keep Rocks from Rolling In.
5. Axe.
6. Pick.
7. Blasting Log Chain.
8. Wrench for Screwing in Guide Lag Screws.
9. Timber Clevice.
10. Drop Pin and Chain to go under Cage.
11. Hanging Bolt.
12. Sand Pump, 0.75-in. pipe, and made in different lengths.
13. Chain Ladder.
14. Guide Clevice.
15. Hanging Bolt Wrench.
16. Wooden Plugs for Drill Holes.
17. Wooden Tamping Stick for Powder.
18. Blow Pipe, 0.75-in. pipe, and made in different lengths.

For cleaning the holes sand pumps are used. (Plate XII.) These are made of ¾-in. pipe with a bend near the end. The shaftman places his hand over the end as the pipe is raised, causing a suction, taking it away as the pipe is pushed down. When this is done rapidly it lifts the sand out of the bottom of the hole. The sand pumps are made in different lengths and sizes to suit conditions. Blowpipes are used for blowing out the holes before blasting, and are usually made of ¾-in. pipe, so that small rocks will have a chance to pass.

As to the placing of holes, Plate VII will show the general method, as there are no ironclad rules. If the ground is breaking out too far on one side the holes are drawn in, or perhaps one may be cut out altogether. It is generally arranged so that each machine has the same number of holes to drill, and this often causes rivalry among the different runners. Both 40 per cent. and 60 per cent. gelatine powder are used, the 60 per cent. being usually used in the cut holes only. In some shafts the whole round is blasted together, but, with 10-ft. rounds, generally the cut holes and one or two rows on either side are blasted first; thus making sure the round will do its work. The cuts are sometimes mucked clear to bottom and the bottom of the cuts blasted with the rest of the round. At other places it is customary to muck down far enough to see that all of the holes have gone and then shoot the rest of the round. The men in this way are twice on the top of a fresh muck pile, and only pick bottoms once to a 10-ft. round. Sometimes a set of timber is placed after the cut holes are blasted and then another after the rest of the round, making both very nicely from the top of the muck pile.

The jackhamer drill is finding its application in certain kinds of rock. The air is blown out through the hollow drill, keeping the holes free from cuttings; in soft ground the holes are likely to plug. At the Pittsmont mine a hole is drilled in the steel sidewise about 1 in. from the lower end of the drill, so that, if the hole in the end becomes clogged, the air blows out through the side, keeping the hole clean above the last inch of the drill. In soft ground the drill bits are enlarged to take 1.25-in. powder. In hard rock, if the bits are enlarged sufficiently to make the hole large enough for 1.25-in. powder, the drilling becomes too slow, while the ⅞-in. powder will not break to the bottom cleanly, and then picking bottom is a slow process. The application of these drills seems to be in rock which is neither too hard nor too soft. A heavier drill of this type might overcome these difficulties if it bored a hole large enough for 1.25-in. powder fast enough in hard rock.

The fuse is doubled for blasting the holes; two nails or hooks are set the right distance apart and the fuse is wound tight around them, and then cut at one of the nails. Caps are put on both ends of the fuse; the crimp in the fuse, made by the other nail, marks the place where the fuse is

Granite Mountain

2'-6"

6'

5'

Nine rows of three holes each

Rainbow

1'

5'

Cut holes

Six rows of four holes each

Badger State

1½'

6'2"

1½'

5'

Cut holes

Ten rows of three holes each
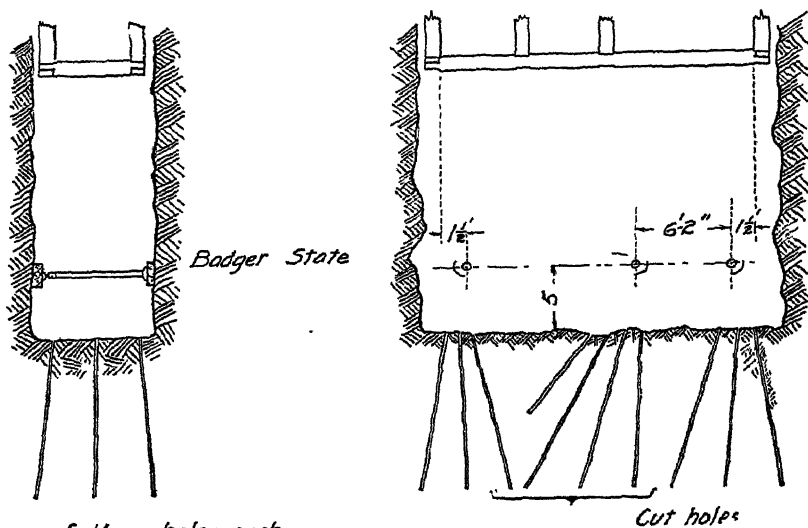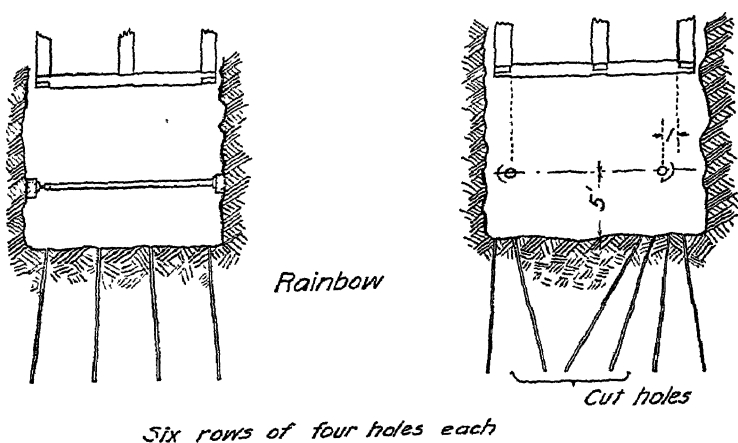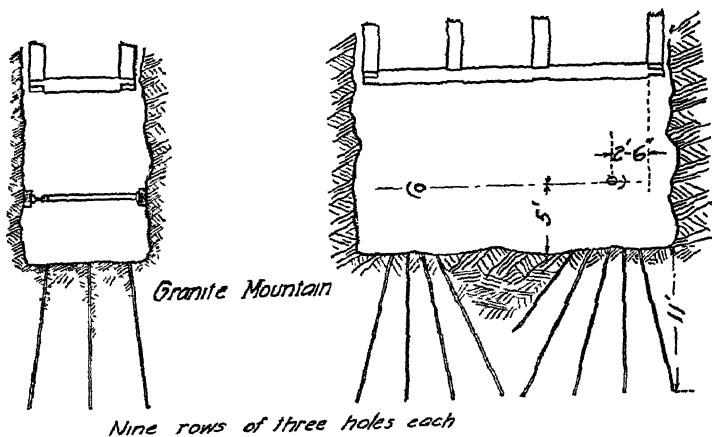
PLATE VII.—PLACING OF DRILL HOLES IN SHAFT.

to be spit, and the fuse is not cut until that particular fuse is spit.   The fuse for the different rows of holes is not cut in different lengths, since the work of spitting gives plenty of relay between fuses.

In the larger mines sinking takes place under a working shaft, the shaft being bulkheaded under the hoisting compartments and several hundred feet above under the auxiliary compartments.   A lining of 2-in. or 3-in. plank is run, separating the auxiliary and main hoisting shafts between these two bulkheads.   The sinking engine is then placed so it may be served by the main hoisting cages.   This bulkhead is often made by crossing the shaft with several layers of 10 by 10 in. timber, removing the shaft lagging for several sets above and filling it with rock.   Sometimes a timber bulkhead is placed, as in Fig. 4, Plate VIII.   A good method is shown in Fig. 2, Plate VIII.   A block of ground is left under the hoisting compartments for about 30 ft., when the shaft is lengthened out to the full.   When sinking is finished this portion is raised, completing the shaft.   This may be seen at the Badger State and Mountain Consolidated mines of the Anaconda company.

Handling Shaft Water.

Some of the shafts of Butte, where the ground has been drained, are being sunk without pumps.   In this case the water is bailed into the sinking buckets with pails.   The bucket is then hoisted and dumped into a car.   The car can be run out away from the shaft, and the plug pulled out; it will empty itself by the time the next bucket of water is ready to be hoisted.

The 10-7-5-10 Knowles and the 9 b Cameron pumps are generally in use in sinking work.   It is becoming the practice, where pumping with compressed air, to force the exhaust air back into the water column.   A check valve is used to keep the water from running back into the cylinder of the pump when the pump is shut down.   A valve is placed so that the air may be exhausted into the atmosphere.   A valve should be placed in the water column, to allow the column to be drained in case the water pressure is too great to allow the pump to start.   A pump so rigged, but for a short lift, and without the column valve, is shown in Plate XIII. It is shown with the 6 to 4 in. nipple on the suction, to make the flanges standard, and with the expansion joint on the water column.

Pumps run in this way are noiseless and will not freeze in a cold shaft because the air leaves the cylinders of the pump still under pressure. The advantages to be gained by this method are that the column is not under such heavy pressure and is less liable to breaks, and the pump will give less trouble with the packing.   A 200-ft. lift can be made very nicely with a 10-7-5-10 Knowles pump.   The column pipes are flanged in 10-ft.
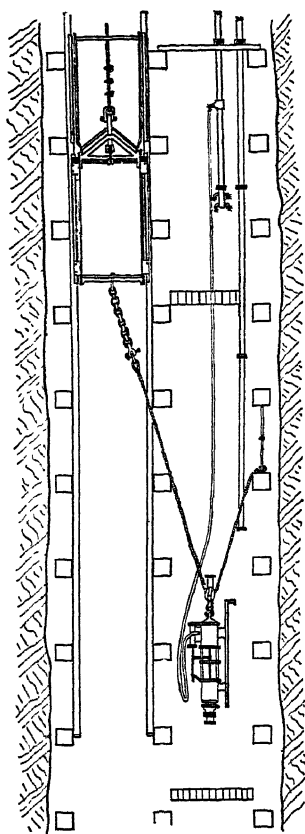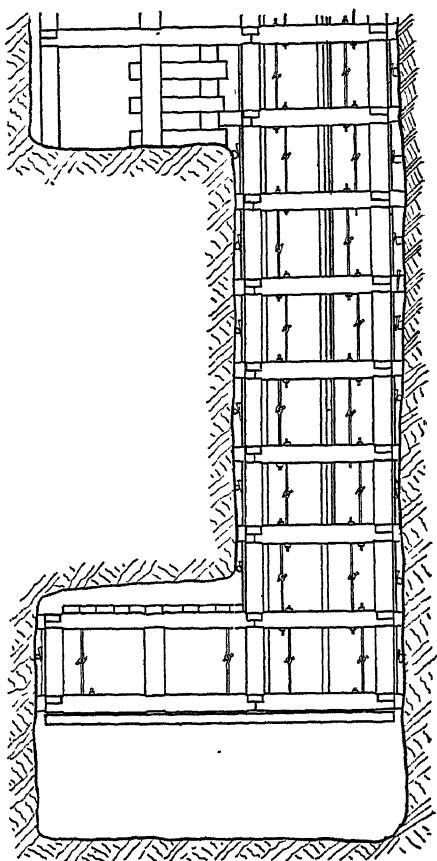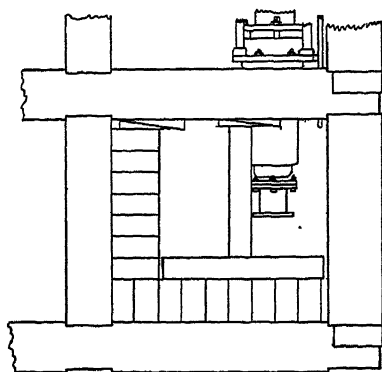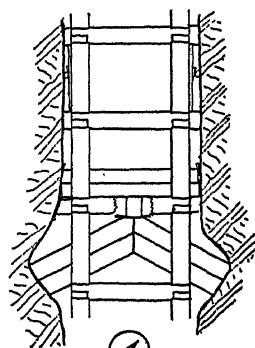
PLATE VIII.

1. Lowering pump with rope.
2. Leaving a block of solid ground under the hoisting compartments of the shaft.
3. A bulkhead under the pump.
4. End view of a timber bulkhead under the hoisting compartments.

lengths, and care should be taken to keep the flanges away from the wall plates. They should be started and held vertical about half-way between sets. Expansion joints should be placed every 200 ft. in the water and air lines. The pump has an expansion, as shown in Plate XIII, which is lowered with it. This takes care of any variation in the 10-ft. lengths
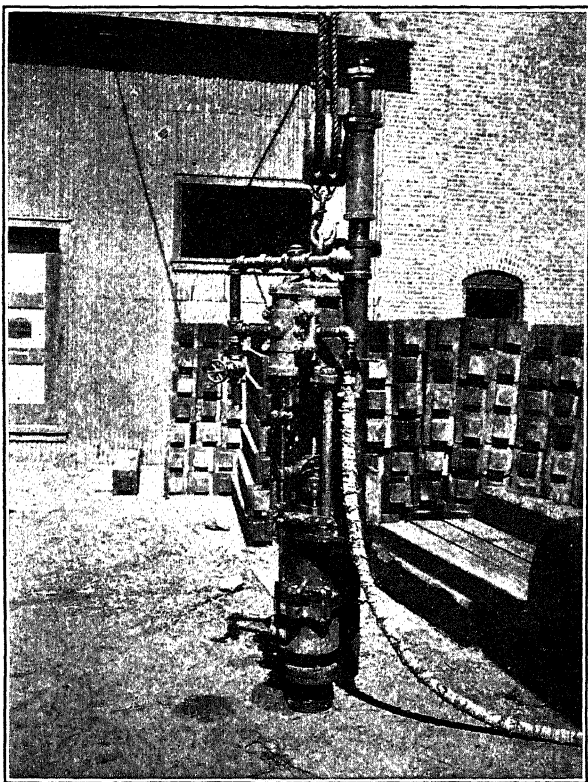
PLATE XIII.—10 BY 7 — 5 BY 10 IN. KNOWLES PUMP. CONNECTED TO THROW EXHAUST AIR INTO WATER COLUMN.

of pipe. The clamps on the water column should be kept 30 to 40 ft. above the pump. In case the pump is not lowered to a position exactly under the pipe, the pipe can be swung over to the pump.

There are several methods of lowering pumps, the favorite one being with chain block. The chain block is fastened to a cross piece several sets above, which is carried down along with the chain block. A hand-operated winch, set in the pump shaft usually at the first station, and a rope leading down to the pump, are used. A separate bell line gives the man at the winch the signal to hoist or lower the pump. The pump may be lowered with a large-size Manilla rope, as shown in Plate VIII,

Fig. 1. This method is very fast. The rope is wound around the timber and tied after the pump is lowered, so that, if the pump is lifted off the wall plate by the blasting, it will not fall to the bottom.

To keep the pump from being blasted a bulkhead is carried along and kept under the pump. As these bulkheads have to be passed down by
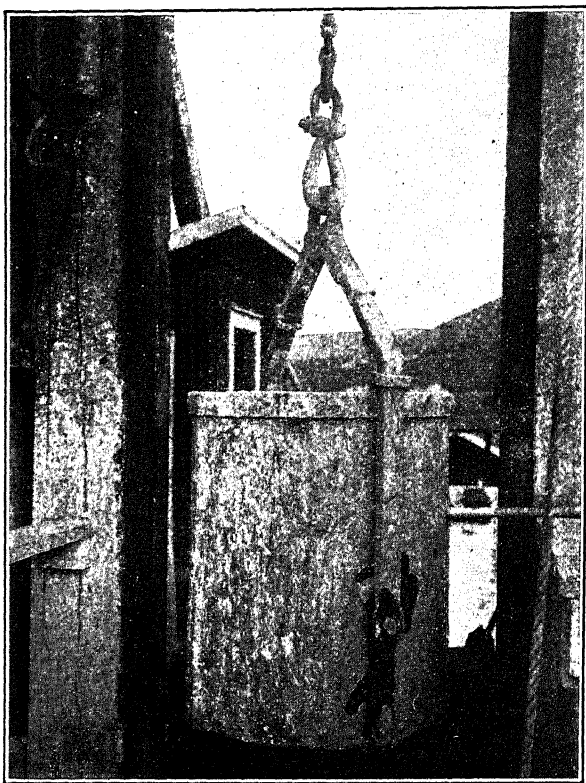
PLATE XIV.—BUCKET AND HOOK ARRANGEMENT FOR SUPPORTING BUCKET.
TROPIC SHAFT, ANACONDA CO.

hand every time the pump is lowered, the timber should be in as light pieces as possible, and 5 by 10 in. or 4 by 10 in. set on edge makes a nice size for this purpose. The bulkhead is designed by the pumpman to suit existing conditions. An end view of a typical bulkhead is shown in Plate VIII, Fig. 3. It is wedged tight in place, and, after the blast, it is loosened so that it may be easily taken out.

The suction of the pump extends through this bulkhead to the sump below; consequently the bulkhead is made so that it can be closed for each blast. Before the blast the suction is disconnected and pulled up above the bulkhead by a double block and tackle. One way to make

this hole for the suction is to have the pieces below the pump wedged so they can be pulled up and out; or they may be kept loose so they can be spread. Another method is to take two 10 by 10 in. timbers and cut a 5-in. hole in each, so that, when they are placed under the pump, they make a 10 by 10 in. hole for the suction. During a blast a block is placed
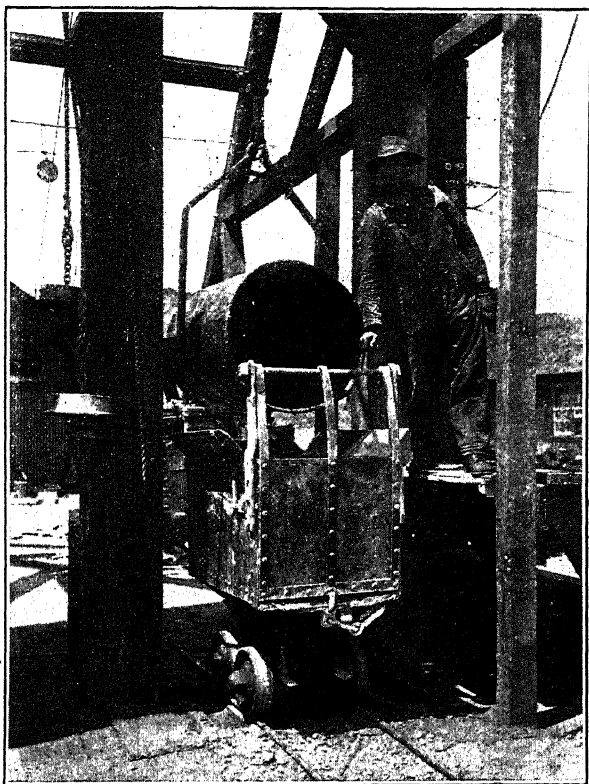
PLATE XV.—DROP PIN METHOD OF SUPPORTING BUCKET. RAINBOW SHAFT.

over the hole and under the pump, to prevent rocks from flying up. In some places the suction is pulled just high enough to leave the strainer in the hole. The strainers for the suction hoses are made at the mines and consist of a piece of pipe of the same diameter as the suction, about 14 in. long, bored full of $\frac{1}{4}$-in. holes, with thread on one end, and flattened together on the plain end. Timbers 10 in. square are very heavy when they are waterlogged, and most pumpmen prefer the first method. The suction hoses are usually wound with $\frac{1}{4}$-in. Manilla rope, as in Plate XI, to prevent wear. Short lengths of flanged pipe are kept with the pump, and are placed between the suction hose and the pump as the rock is being mucked down. As the suction hose can be lowered with the block and

tackle, this is an easy matter. A leak in the suction is a great annoyance, and about the only way to find it is to place the ear near the flanged joints and listen for the sound of the leak.

Rock Handling.

Sinking engines have every possible speed, depending upon the depth they have to work. The type of cage shown in Plate XVII is generally
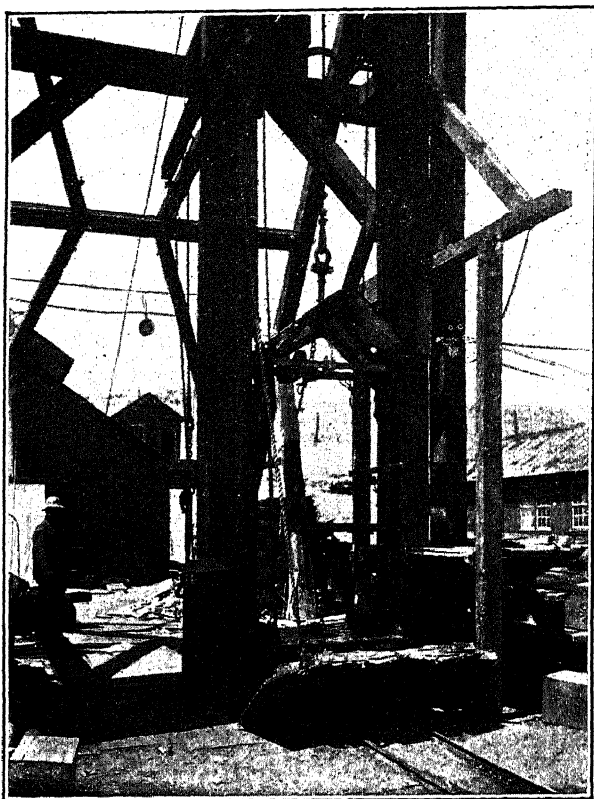
PLATE XVI.—CAGE AND SINKING SHOES.   RAINBOW SHAFT.

used for sinking; the long shoes along the side and the sinking shoes above can be seen. The corner bars of the cage are drawn in so that men can step off the cage on to the timber. In Plate XVII a regular mine cage is fitted for sinking, having sinking shoes. The sinking bucket with straight sides has come into almost universal use. These buckets are about 2 ft. 6 in. across the top, and 3 ft. high, holding one mine car of rock. Barrel-shaped buckets and buckets wider at the top than at the bottom have not found much favor. The bucket must be wide at

the bottom so that it will sit steadily on the muck below, buckets with small bottoms being likely to topple over and hurt the men.   There are many novel methods of securing the bucket; the best and most reliable one, in my estimation, being shown in Plate XV, which is known as the drop pin.   Another method is shown in Plate XII, Object 2, this being
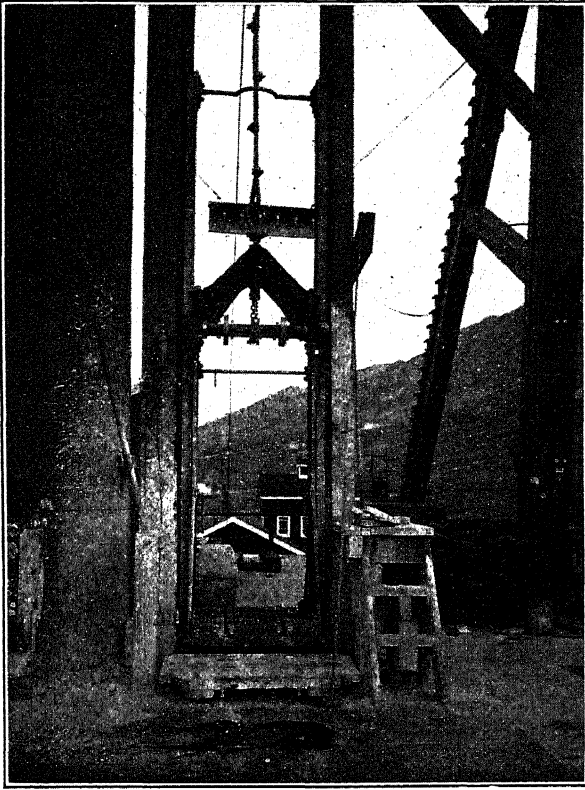
PLATE XVII.—USUAL FORM OF SINKING CAGE AND SINKING SHOES.
TROPIC SHAFT, ANACONDA CO.

a link flattened for about 0.75 in. on one side, and which can be passed into the hook shown, the chain being passed around the bail of the bucket. Plate XIV shows two hooks turned in opposite directions, this system is being used at the Tropic shaft of the Anaconda company.

    The cars should be large and have strong bottoms.   Large boulders are lifted into the bucket by means of a rock chain attached to the cage, and dumping these into the cars soon breaks them if they are light. Plate XV shows a common mine car with raised door, so that large boulders may be dumped out of it.   A rock chain may be seen in Plate XII. It is well to have several of these made in different weights of chain, and

0.5-in. chain is a good average size for this purpose. A chain ladder is also shown in Plate XII, this being made of 0.5 by 18-in. bolts, with ⅜-in. chain and 0.75-in. pipe cut to go over the bolts between the chains. The ends of the bolts are riveted after the nuts are in place; in case a step is broken it is easily replaced. The chain ladder should be hung so that, in case anything should go wrong with the sinking engine, the men can climb up the chain ladder and out. In some places this is lengthened to about 35 ft., and the wooden ladders are kept down to connect with the chain ladder.

Miscellaneous Details.

Where heavy ground occurs in the shaft, and the timber is being pushed in and broken, jacket sets are placed outside the regular shaft timbers. (Plate IX.)   This gives an opening around the shaft, where men can work easing off the timber while the shaft is in operation; these jacket sets are raised alongside the shaft, although in a few instances they have been carried down with the sinking. The four small compartments at the corners are used as chutes, and discharge at the station below. The rock is generally very fine, and is picked out in easing off the timber. If any of the outer jacket timbers break they can be easily replaced without disturbing the shaft.

Sometimes the shaft sets themselves are pushed out of line. Vertical stringers are then run up and down the shaft, being wedged to three sets all around. The jacket timber of the middle set can be·then wedged and held from coming in when the blocks between them and the shaft are taken out. The shaft set is then pushed over with a jack bar to its right position and new blocks are fitted to go between the shaft set and the jacket set. This is then repeated on the next bad set. In raising alongside the shaft for the jacket sets, a set is finished around the shaft before commencing another. The place where the ground is the heaviest is selected as the starting point for the raising; what natural support the better ground can give is thus taken advantage of.

For signals, a 0.75-in. Manilla rope, attached to a bob, is generally used, and the shaftman ties a loop on the end of the rope to make it easier to pull. When the men are being hoisted, after spitting the holes, this loop is hooked on the cage and the rope is thus hoisted away from the blast. Rope bells can be used successfully for 2,200 ft. in depth, but, after that, a combination of electric bell and rope bell must be resorted to.

Electric lights are a great aid in shaft sinking, as they leave the men unencumbered with candlesticks, etc. Especially is this true in drilling, when the exhaust of the machines makes it hard to keep a light of any kind. When timbering, candles are used to give light back of the timber. An old reel may be used to hold the light wires. A couple of old pieces of

shovel handle nailed about 5 ft. apart may be used for the same purpose. To protect the lights from rocks falling from above, a heavy reflector is kept above them. Electric lights are particularly useful where electric
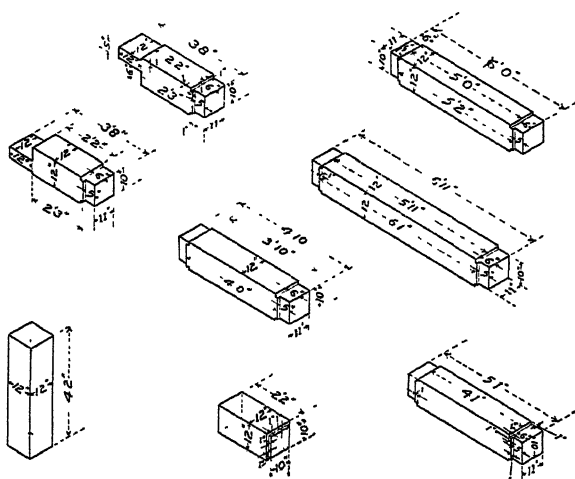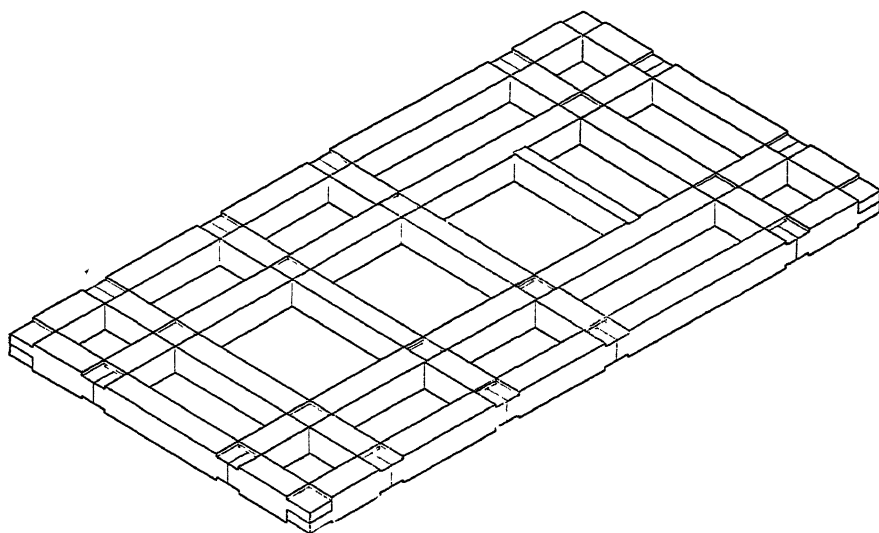
PLATE IX.—JACKET SETS, TRAMWAY MINE.

engines are used in sinking, as they give notice to shaftmen of any interruption in the current.

To ventilate the shaft, 1-in. lining boards are run down, separating two compartments of the shaft. These boards are cut to go between centers, and cleats are nailed on both their sides, at top and bottom, to hold them

in place. In order to make a circulation, a ventilating stack, called an upcast, is placed at the top of one of these compartments running up alongside the gallows frame. Where the shaft is to be sunk under hoisting shafts, the ventilation is much harder and blowers are sometimes used to furnish air. The time it takes smoke to clear away is time lost in shaft sinking. In some places the cages are run up and down through the shaft after blasting to help clear away the smoke.

DISCUSSION.

GEORGE A. PACKARD, Butte, Mont.:—I would like to ask Mr. Braly if he has any information as to the relative efficiency of the sinking pump, using it as he refers to it, discharging the exhaust into the water column or otherwise.

MR. BRALY:—I have never gathered any information on that point. It is a question of getting the work done. In driving the shaft, using that method, we find it easier on the men and everything else. In the bottom of the shaft, if you exhaust back into your column, it is a quiet shaft. Another thing is the freezing: if you are sinking with air in your pump, you will find it is liable to freeze, and in that case as the air is still under compression it will not freeze the pump. The difference is very small if there is any. But, as might be expected, the pump is a little less efficient when exhausting the air into the column.

MR. PACKARD:—I suppose all of us know what a pump in the bottom of the shaft is. My experience has been, although I have no figures, that the capacity of a pump is quite considerably decreased if you are pumping, say, from a depth of 200 ft. through a No. 7 Knowles pump, when you exhaust the air into a column, but, on the other hand, it is also my impression that we can raise the water to a little greater distance. I would like to get some figures from somebody who has made actual tests.

The Precipitation of Copper from the Mine Waters of the Butte District.

BY J. C. FEBLES, BUTTE, MONT.

HISTORY.

THE use of iron for the precipitation of copper was known at least as early as the fifteenth century. Both Paracelsus and Basil Valentine refer to it in their writings, as early as 1500 A. D. It was used in Peru for this purpose prior to 1637, and at Rio Tinto, Spain, as early as the sixteenth century.

The statement has been made repeatedly that in the early history of the mining industry at Butte, the practicability of this method of recovering copper from solution was overlooked, and that enormous quantities of copper-bearing water were allowed to go to waste, the copper in which might easily have been recovered at small cost; and that this oversight on the part of the mine operators was due to ignorance of this method of recovering copper. As a matter of fact, very little copper-bearing mineral was encountered in the mines of Butte until a depth of 400 or 500 ft. had been reached, and in many cases much greater depths were necessary to reach the copper-bearing zone.

The water encountered in the upper levels (although in most cases quite highly mineralized) contained little or no copper, and it was only after considerable areas of copper-bearing ore had been uncovered and developed that copper sulphate began to appear in the mine water in any considerable quantity. In fact, the first water that was observed to contain copper carried it in such small amounts as to be of little or no importance. It was only after years of underground work had opened extensive areas of stopes, drifts, and other underground workings, these being subsequently filled with waste, nearly all of which contained small copper values, which became oxidized by exposure to the air and were acted upon by the slowly percolating mineral water, that the recovery of copper from mine water became commercially important.

Below is given the analysis of a sample of mine water taken from the Gagnon mine, analyzed by W. F. Hillebrand.[1] Figures are parts per million.

SiO_2	29.5	Fe_3	None
Ca	512.1	Mn	1.4
Mg	102.6	Cu	Trace
K	11.4	PO_4	Trace
Na	82.8	SO_4	593.4
Fe_2	0.4	Cl	842.8

Although this water is not copper bearing, it is not hard to see that it could easily become so if brought in contact with soluble copper minerals.

At this time the Gagnon mine was pumping about 150 gal. of water a minute, and some time subsequently a careful assay of the water showed 0.0016 per cent. of copper, which would be equivalent to about 28 lb. per day in the water pumped. An attempt to recover copper from such water would not have paid the cost of maintenance of the precipitating plant.

When mining operations had reached such a stage as to allow such water to come in contact with copper salts, which, from oxidation, had become soluble; then, and not until then, mine water began to carry enough copper in solution to make its recovery profitable.

The first production of copper precipitate from the mine water of Butte was from the St. Lawrence mine. In November, 1889, a fire broke out on the 400 level of the St. Lawrence, and an effort was made to extinguish it by filling the mine with water, but, owing to connected workings of other properties, it was found to be impossible to make the water rise high enough to submerge the fire zone. When this water was subsequently pumped out of the mine it was found to be very rich in copper sulphate. It has been stated that when pumping first began this water contained as much as 0.75 per cent. of copper. The statement cannot be verified, however, at this time.

When pumping operations were started for the purpose of unwatering the St. Lawrence the water was allowed to flow down the hillside and find its way to the gulch below. In doing this it flowed through the yard of a man named Miller, who was at that time living in East Butte. This Mr. Miller enlarged and cleaned out the ditch which carried this rich copper-bearing water through his yard, placed scrap iron and tin cans in the ditch, and thus began the pre-

[1] W. H. Weed : Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey* (1912).

cipitation of cement copper. Subsequently he improved his little plant by building some boxes and flumes of old lumber. He sold precipitate, in small lots, to the Colorado Smelter. His operations were of short duration, however, and were carried on only upon a very small scale.

In 1890 William Ledford secured from the Anaconda Copper Mining Co. a lease on the water pumped from the St. Lawrence mine. He began operations by digging holes in the ground, which were filled with scrap iron and tin cans, and allowing the water to flow through these holes and over the iron. After operating in this way for a time he built a flume of lumber, about 40 ft. in length, which he used in conjunction with the ground holes. The results obtained with the flume were evidently satisfactory, for he continued adding to this flume until he had a plant of sufficient size to enable him to abandon precipitation in the ground holes.

Ledford's operations were profitably carried on during the continuance of his lease. He sold his precipitate to both the Colorado and the Parrot Smelters.

At the expiration of Ledford's lease, the Anaconda Co. decided to itself operate the precipitating plant, which it has continued to do up to this time.

As the development of other mines became more extensive, furnishing air passages for the oxidation of low-grade copper sulphides in stope filling and other places, percolating waters were enriched in copper sulphate to such an extent that it became profitable to erect and operate precipitating plants for its recovery. Consequently, within the next ten years, most of the larger mining companies were profitably operating such plants.

PRINCIPLES OF THE PROCESS.

As is well known, the principal reaction which occurs is the precipitation of metallic copper at the expense of metallic iron, by galvanic action, and may be expressed by the equation,

$$Fe + CuSO_4 = FeSO_4 + Cu.$$
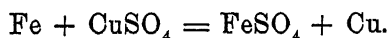
The resulting ferrous sulphate becomes oxidized by constant aeration in the flumes and towers to ferric sulphate, which, being subjected to still further oxidation, is thrown out of the solution, both as basic ferric sulphate and as ferric hydrate, with the liberation of free acid, which reacts upon either copper or iron salts, again forming the corresponding sulphates.

The quality as well as the quantity of the cement copper obtained in a given time depends upon a number of conditions, which may be briefly enumerated as follows :

1. The amount of copper in solution.
2. The temperature of the solution.
3. The freedom of the solution from salts other than copper.
4. The rate of flow.
5. The kind of precipitating apparatus used.
6. The kind of iron used.
7. The size and shape of the iron, and the way it is placed.
8 The proper handling and cleaning of the iron.

1. It will require no argument to make plain that the greater the percentage of copper sulphate the solution contains, the more rapid will be the rate of precipitation, and also the better the quality of the precipitate.

2. The greater the temperature of the solution, the more rapid the deposition.

3. The freer the solution is from iron and other sulphates, the more rapid the deposition and the better the quality of copper obtained.

4. Within reasonable limits, the more rapid the flow the more satisfactory the precipitation, both in quality and quantity. The most modern practice is to construct the plant with the greatest amount of fall to the first flumes in the system, in order to secure a rapid flow at first. The fall is gradually diminished until, at the end of the operation, the water is flowing quite slowly.

5. There has been much argument as to whether towers or flumes will give the best results; but after long-continued use of both and close comparison, the conclusion is that the flumes are superior to the towers in almost every way. The first cost of the flume is less; it can be more easily and economically handled; the cost of maintenance is less; the yield of copper per unit area is greater, and the quality of the precipitate is just as good.

The tower is much more difficult to handle, and in cold weather is almost impossible to care for properly, owing to the formation of ice in large masses in such positions as to make its removal difficult, and in some cases impossible.

In fact, about the only advantage that can be claimed for the tower is, that it permits the use of large and irregular pieces of iron, which cannot be used to advantage in flumes. For example, old boilers, roll shells, trommels, and such cumbersome material can be well

utilized in a tower, while their economical use in a flume would -require cutting or breaking, adding expense to handling.

6. Almost any merchant iron or mild steel will give satisfactory results, provided its surface is kept clean. The results obtained by the use of very hard steel, or spring steel, are not so good. Cast iron is the poorest of all, having little or no value for precipitating. The uncombined carbon in the cast iron seems to interfere, to a very marked degree, with the reaction. The copper adheres very strongly to the surface of the cast iron, making its removal so difficult that the surface cannot be properly cleaned. After the copper coating has become quite thick it will break off in hard flakes and scales, with pieces of carbon adhering to them.

The best iron for the purpose seems to be old rails, merchant bar, and pipe of small diameter. Horse shoes, punched screen, heavy sheet iron, nails, bolts, and iron wire are very good; roofing iron, tin cans, and similar scrap are not so good, but are extensively used ·because they are plentiful and cheap. All paper should be removed from cans before using. Painted iron and galvanized iron should be burned before using; in fact, annealing at a low temperature improves almost any iron for the purpose of precipitation.

7. Care should be used in the distribution of the iron, in order to secure the maximum precipitating surface. It should be packed quite closely, but in such a manner as to allow the water to flow freely. Rails, pipe, and bar iron should be placed horizontally in the flume, with short transverse pieces between the layers, to allow space for the free flow of the solution. Material of this kind can be used to the best advantage in the upper part of the system, where the flow is most rapid and the water richest in copper. Smaller scrap and cans may be placed indiscriminately. Flat sheets should be placed one above another, with enough small scrap between to prevent interruption of the flow, and in such a way as to present the maximum surface. Large and heavy pieces are used to the best advantage in the towers.

8. Where rails, small pipe, and bars are used, and the flow is sufficiently rapid, very little or no handling of the iron is necessary, an occasional sweeping with a stiff brush being sufficient. The precipitated copper, thus freed from the iron, will be carried to the settling tank below.

The smaller scrap, sheets, wire, and cans, should be turned over occasionally, with hooks and rakes, in order to remove the precipitated copper.

The iron in the towers requires very little attention, as in most

cases the fall of the water will keep the surface of the iron clean and free from copper.

Where the grade of the flume is not sufficient for the water to carry the precipitate to the settling tank below, the flume is cleaned up at intervals by removing the iron (first separating all the copper possible from it) and the copper is sluiced into the settler. As these flumes are constructed in two or more parallel sections, the water can be handled through one section while another is being cleaned up.

After the expiration of Mr. Ledford's lease, the Anaconda Co. continued the precipitation of copper at the St. Lawrence mine until 1901. About this time the High Ore pumping plant was completed, which enabled the company to handle the water of the entire group through the High Ore shaft. A new precipitating plant was then built, convenient to the High Ore. Since then practically all of the Anaconda Co.'s precipitating operations have been carried on at this plant.

About the time the present plant was started (1901 or 1902), some experiments were made for the purpose of precipitating the copper by neutralizing the water with lime. The Gagnon mine used this method of neutralizing the mine water, in order to protect the pumps and columns from corrosion by the acid in the water.[2]

The use of lime in this way, at the Gagnon, probably suggested its use for the recovery of copper from mine water. This was quickly abandoned, however, owing partly to the low copper value of the resulting precipitate and partly to inability to handle the precipitate economically. The precipitate is very light and flocculent, in consequence of which it settles very slowly, and is so sticky and gelatinous that it cannot be filtered.

The following shows the result of an analysis of a precipitate obtained by adding a 10 per cent. excess of quicklime to a sample of the High Ore mine water. The sulphuric acid was first determined and the proper amount of lime to combine with it was calculated, 10 per cent. in excess of this amount being added:

	Per Cent.
Copper,	8.12
Iron,	10.08
Sulphuric acid,	8.23
Alumina,	4.53
Lime,	15.80
Magnesia,	0.60

The filtrate from the above precipitate, upon being tested, was found still to contain sulphuric acid. Upon the addition of more

[2] U. S. Patent No. 567,312, Sept. 8, 1896 ; C. W. Goodale and H. W. Hixon.

lime, however, nothing precipitated but the sulphates of lime and magnesia, and they came down very slowly.

By adding a very large excess of lime and allowing to stand several hours, a filtrate can be obtained which is practically free from sulphuric acid. Analysis of a precipitate obtained in this way is given below :

	Per Cent.
Copper,	3.20
Iron,	2.00
Sulphuric acid,	3.50
Alumina,	0.71
Lime,	61.20
Magnesia,	2.16

Even where the precipitation is carefully made, and a large excess of lime avoided, the precipitate obtained in this way is of little or no value, for the reasons stated above.

THE HIGH ORE PLANT.

The water is pumped by steam and electric pumps to the 300-ft. level, where it runs by gravity through a tunnel, at the outlet of which it enters the first set of precipitating flumes. There are three of these, placed parallel to each other. They average about 4 ft. in width and are 2 ft. deep. Their length is about 300 ft. to the first settling tanks, the average grade being 2 per cent.

These flumes are filled with selected scrap iron, such as rails, pipe, rods, and bars, and such other iron as will give a rapid, clean precipitation, this heading water being comparatively rich in copper. Under normal working conditions about 1,200 gal. of water per minute is delivered into these flumes, containing about 0.0500 per cent. of copper. The temperature of the water as it enters the plant is approximately 85° F.

The iron in these flumes is turned over and handled constantly, in order to separate the precipitated copper and leave a clean iron surface for further precipitation. Every few days the water is shut off from one unit, allowing the remaining two units to carry all, so that the iron may be temporarily removed, while the precipitate is washed into the settler below. This work is done in rotation, so that all flumes receive an equal amount of attention.

After passing the first settling tanks the water continues in the flumes as described above, the only change being that the grade is increased to about 2.5 per cent. and the flumes are increased in width to about 8 ft. Settling tanks are placed approximately 75 ft. apart

from this point for a distance of 500 ft., which, with the 300 ft. of flume above the first settlers, constitutes the first unit of the plant.

These settling tanks are merely large wooden boxes, of the same width as the flume, about 15 ft. long and 8 ft. deep, and so arranged that the precipitate can be sluiced through a hole in the bottom into a trough connected with drying vats which are conveniently located; the tops of the drying vats being somewhat lower than the bottoms of the settlers.

This first unit, which consists entirely of flumes, is approximately 800 ft. in length, and has a total fall of 30 ft., or an average grade of 3.75 per cent. The average velocity of the water in this portion of the plant is 60 ft. per minute.

The water, after leaving this first set of flumes, again passes through settlers, from which it is discharged into the first tower, which is merely a heavy wood frame, the main supports of which are 8 by 8 in. square timbers. It is filled with pieces of scrap iron which are too large, or for some other reason are unsuitable, for use in the flumes. Baffle boards are nailed to the sides and ends, to prevent the falling water from splashing outside the tower. The dimensions of this tower are 130 by 8 ft. by 19 ft. high. with a settling tank beneath which extends the entire length.

A summary of the work accomplished by the above described flume and tower is given below:

Temperature of water entering flume, ...	85° F.
Temperature of water entering tower, ...	78° F.
Temperature of water leaving tower, ...	70° F.
Per cent. of copper in water entering flume, ...	0.0516
Per cent. of copper in water entering tower, ...	0.0027
Per cent. of copper in water leaving tower, ...	0.0021
Per cent. of copper extracted by flume, ...	95.03
Per cent. of remaining copper extracted by tower, ...	22.20
Per cent. of copper extracted by flume and tower, ...	95.93
Total time of water in flume, ...	18 min.
Average per cent. of copper in precipitate from settler above tower, . .	75.20
Per cent. of copper in settler under tower, ...	9.48

The water then passes through another tower, from which it enters two large parallel flumes, 9 ft. wide by 300 ft. long. This double flume is filled with tin cans and small scrap. The total fall of this flume is 4 ft., or a 1.3 per cent. grade. The velocity of the water at this point is about 50 ft. per minute, the time consumed in passing through being about 6 min. The water entering this second flume assays 0.0019 per cent. of copper, and leaving, 0.0017 per cent., the extraction in this flume being 10.52 per cent. The precipitate made by this flume contains 12.50 per cent. of copper.

The water, upon being discharged from this flume, passes through five other towers, the first of which is a large one, being 81 by 11 ft. by 25 ft. high. The remaining four are smaller, averaging only 8 or 9 ft. in height. From the last of these towers the water enters a settler 27 by 28 ft. by 4 ft. deep, from which it is allowed to run to waste.

Following is a summary of the work done by the last four towers:

Per cent. of copper in water entering, ... 0.0017
Per cent. of copper in water at outlet, ... 0.0010
Per cent. of copper extracted by last four towers, ... 41.2
Total extraction of plant (per cent.), ... 98.6
Temperature of water at outlet, ... 54° F.
Per cent. of copper in precipitate from last settler, ... 8.60
Per cent. of iron oxide in precipitate from last settler, ... 47.71
Per cent. of alumina in precipitate from last settler, ... 8.69
Per cent. of silica in precipitate from last settler, ... 7.60

The following table shows the result of a complete analysis of both head and tail water of the High Ore precipitating plant, arranged in such a way as to be easily compared:

	No. 1. Heading Water.	No. 2. Tailing Water.	Gain +or Loss— of No. 2 as Compared with No. 1.	No. 1. All Bases Calculated to Normal Sulphates.	No. 1. SO_3 Required.	No. 2. All Bases Calculated to Normal Sulphates.	No. 2. SO_3 Required.
SiO_2	0.0504	0.0384	— 0.0120	0.0504	. . . .	0.0384	. . . .
CuO	0 6683	0.0125	— 0.6558	1.3405	0.6722	0.0251	0.0126
Al_2O_3	0.2102	0.1926	. . . .	0.7046	0.4944	0.6452	0.4526
Fe_2O_3	0.5931	0.0729	. . . .	1.4845	0.8914	0.1826	0.1097
FeO	0.2194	1.0427	. . . .	0.4637	0.2443	2.2035	1.1608
MnO	0.0714	0.0574	— 0.0173	0.1519	0.0805	0.1151	0.0610
ZnO	0.6969	0.4679	— 0.2290	1.3823	0.6854	0.9281	0.4602
CaO	0.3224	0.3296	+ 0.0072	0.7825	0.4601	0.8000	0.4704
MgO	0.1200	0.1376	+ 0.0176	0.3580	0.2380	0.4106	0.2730
SO_3	3.5246	2.7614	— 0.7632	. . . .	. . . .	. . . .	. . . .
Cl	0.0724	0.0862	+ 0.0138	. . . .	. . . .	. . . .	. . . .
Total, . . .	6.5491	5.1959	. . . .	6.7184	3.7663	5.3486	3.0003

	No. 1.	No. 2.
Total normal sulphates by evaporation (Fe oxidized), ...	6.9856	6.1504
Total sulphates by calculation (Fe oxidized), ...	6.8675	6.0454
Difference (alkali sulphates?), ...	0.1181	0.1050
Fe_2O_3 in suspension as "ocher" (included in above Fe), ...	0.2250	0.0683
(Analysis by Charles M. Palmer.)		

When the settling tanks become filled with precipitate it is sluiced into the drying vats, where, after sufficient time has elapsed for set-

tling, the supernatant water is decanted and the cement copper is
allowed to dry by evaporation until sufficiently compact to be han-
dled. The precipitate, still containing considerable moisture, is
sacked and shipped in car-load lots to the smelter for further treat-
ment. Below is given an average analysis of cement copper:

	Per Cent.
Moisture,	12.2
Copper (electrolytic),	70.9
Silica,	2.3
Oxide of iron (ferrous),	8.4
Alumina,	2.9
Lime,	0.3
Sulphur,	0.8
Arsenic,	0.27
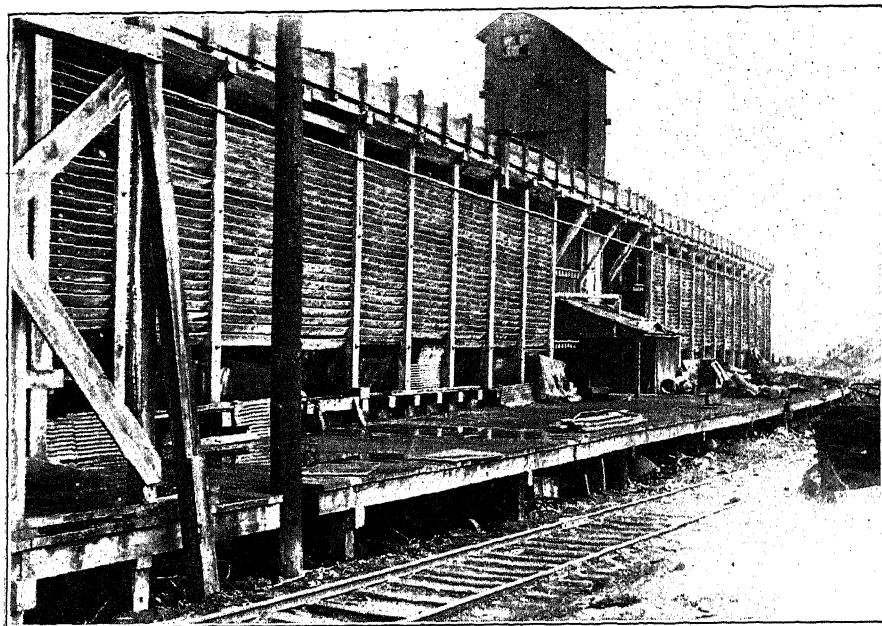
FIG. 1.—LEONARD MINE PRECIPITATING PLANT, BUTTE, MONT.
FRONT VIEW OF TOWER.

Working under conditions similar to those here described, this
plant will produce approximately 2,200,000 lb. of pure copper annu-
ally, which amounts to the shipping of about 10 cars of crude pre-
cipitate each month.

The iron consumed in the process weighs from 1.1 to 2 times as
much as the copper produced. The lower the value in copper, or
the higher the ferric sulphate or free acid, the greater the consump-
tion of iron. Good, clean, compact iron will precipitate more copper
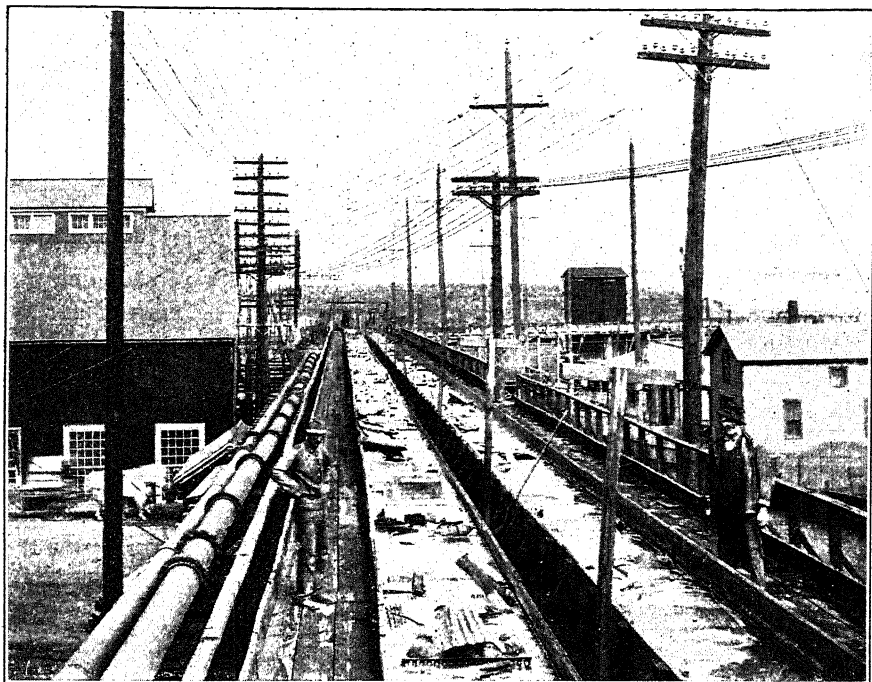
FIG. 2.—VIEW OF LOWER FLUME.
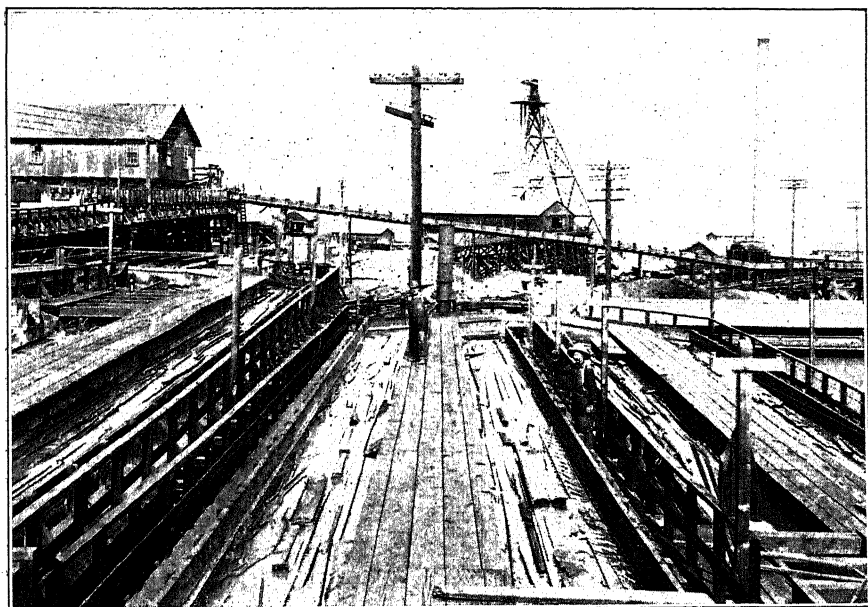
FIG. 3.—VIEW OF UPPER FLUME.

than tin cans and similar light iron.   At the Silver Bow plant, where a large percentage of railroad rails and similar desirable iron is used, 1,700 lb. of copper recovered per ton of iron used is claimed.   A fair estimate for normal conditions would be 1 lb. of copper recovered for each 1.5 lb. of iron consumed.   At the Silver Bow plant it is claimed to be cheaper to use old rails at $11 per ton than cans and miscellaneous scrap at $7.50 to $8.

A full description of the Leonard and, Silver Bow plants would be largely repetition, as in the main they are very similar to the one just described.   Views of the tower and flumes at the Leonard plant are given in Figs. 1 to 3, but the following table will give a good idea of the comparative size and performance of the three plants, these being the principal ones now in operation:

	High Ore Plant.	Leonard Plant.	Silver Bow Plant.
Total length of flumes (excluding towers), feet,	1,769	1,943	805
Total fall, feet, ...	180	87	56
Average fall per 100 ft , feet, ...	10.25	5.6	7
Total height of towers, feet, ...	100	38	20
Total length of towers, feet, ...	630	224	25
Average width of towers, feet, ...	11.45	12.7	7
Total area of towers, square feet, ...	7,213.5	2,315	175
Average width of flumes, feet, ...	12.5	5.7	4.9
Total area of flumes, square feet, ...	22,169	13,400	3,908
Average temperature of water entering, . . .	29°C = 85° F	36° C = 97° F	24° C = 75° F
Average temperature of water leaving, ...	12°C = 54° F	11°C = 52° F	17° C = 62° F
Average flow of water, gallons per minute, . .	1,200	1,500	356
Average velocity in flumes, feet per minute, . .	70.8	61	50.25
Total time of contact, minutes, ...	26.5	29	16
Average copper content entering, per cent., . .	0.0516	0.0482	0.0748
Average copper content leaving, per cent., . .	0.0010	0.0020	0.0082
Copper extracted, per cent., ...	98.6	95.85	90.37
Total length of settlers, feet, ...	938	1,150	128
Average width of settlers, feet, ...	11.31	7.65	9.3
Total area of settlers, square feet, ...	10,613	7,781	1,188
Total precipitating area, square feet, ...	29,382	15,725	4,300

OTHER PRODUCTS PRECIPITATED.

The presence of the ferric hydrate, or " ocher," in the precipitate is objectionable, and is to be avoided as far as possible.   There does not seem, however, to be any satisfactory way to control its precipitation. The formation of this precipitate is due to oxidation of iron in solution, probably by aeration, which results in its being thrown down both as ferric hydrate and as basic ferric sulphate.

The precipitation of this " ocher " is less at the Leonard plant than at any of the others, owing, most likely, to the presence of small particles of organic matter in suspension in the water, which has a

tendency to prevent, or at least to retard, the oxidation of the ferrous salts.

An analysis of the "ocher" from High Ore tail water gave the following results:

	Per Cent.
Silica,	3.11
Copper oxide,	Trace
Alumina,	1.71
Ferric ioxide,	66.72
Zinc oxide,	Trace
Manganese oxide,	None
Lime,	None
Magnesia,	None
Sulphuric anhydride,	11.51
Water,	16.95 (calculated for limonite)

Calculating all SO_3 to basic ferric sulphate $(Fe_2(SO_4)_3 : (Fe_2O_3 + 8 H_2O)$ and the remaining Fe_2O_3 to limonite $(2 Fe_2O_3 : 3 H_2O)$, the result is as follows:

	Grams per Liter.	Per Cent.
Silica,	0.0160	3.11
Alumina,	0.0088	1.71
Limonite,	0.3084	59.95
Basic ferric sulphate,	0.1812	35.23
	0.5144	100.00

The "ocher" was filtered from water which had stood in the laboratory about 10 days, and had been frequently exposed to the atmosphere. The amount of "ocher" obtained was 0.5072 g. per liter, dried at 100° C.

The mine waters of Butte contain small amounts of arsenic, part of which is precipitated with the copper. The average of four weekly samples of Leonard mine water gave the following results for copper and arsenic:

	Copper. Grams per Liter.	Arsenic. Grams per Liter.
Heads,	0.236	0.00125
Tails,	0.017	0.00082

The amount of water pumped during the month was 37,000,000 gal., the copper and arsenic precipitate being:

Copper, 0.0219 per cent., or 68,000 lb.
Arsenic, 0.000043 per cent., or 135 lb.
The ratio of arsenic to copper in the mine water is 1 : 189.
The ratio of arsenic to copper in the precipitate is 1: 509.

This would give 0.135 per cent. of arsenic in a precipitate assaying 68 per cent. of copper.

An analysis of mine water, made at the Great Falls Smelter, gave results as follows :

Grams per Liter.

Free acid,	0.1200
Arsenic,	0.00915
Antimony,	0.00570
Copper,	0.1573
Total iron,	0.2310

This analysis shows that the ratio of arsenic to copper in the mine water was about 1 to 17, and on this basis, if all the arsenic came down with the copper, a product assaying 65 per cent. of copper would run 3.8 per cent. of arsenic. As analysis of the precipitate shows no such proportions, it would appear that only part of the arsenic is precipitated from the solution.

Four samples of precipitate were taken from the High Ore plant, at different places, and sent to the Great Falls Smelter for analysis, in order to show the relation of copper to arsenic precipitated in various places. The results of these analyses are as follows :

	Copper.	Arsenic.
	Per Cent.	Per Cent.
Precipitate from first settler, 300 ft.,	73.54	0.14
Precipitate from settler, 900 ft.,	48.74	0.23
Precipitate from settler, 1,400 ft.,	27.43	0.41
Precipitate from last settler,	8.60	0.45

This seems to show conclusively that the arsenic does not precipitate proportionally with the copper.

Large samples of both head and tail water from the High Ore plant were evaporated, and silver assays made on the residue. Nothing more than traces of silver were obtained from either sample, although a precipitate containing from 70 to 80 per cent. of copper will carry from 0.3 to 0.4 oz. of silver per ton.

Costs.

An average of cost figures, extending over one year's operation of the Leonard plant, is as follows :

Per Pound of
Copper Produced.

Labor,	$0.0352
Supplies and iron,	0.0140
Sundries,	0.0002
	$0.0494

The above figures are upon the basis of the net pounds of copper contained in the precipitate shipped and paid for.

MINE PUMPS AND COLUMNS.

A brief description of the methods employed in Butte for the protection of the pumps and water columns from the acid water may be of interest.

The water ends of all pumps are constructed, as far as possible, of a phosphor-bronze alloy analyzing approximately 85 per cent. of copper and 15 per cent. of tin. Experience has shown that this alloy will stand the corrosive action better than any other material yet tried. Plungers and valves made of this bronze are not altogether proof against the action of the water, however, having to be replaced from time to time. The corroded parts are, upon removal, sent to the machine department for repairs, a large supply of interchangeable parts being kept on hand.

The columns are constructed of cast-iron pipe, lined with either lead or wood. The lead-lined pipes are flanged, the lead lining flaring, so that when bolted together, with a rubber gasket between, a water-tight joint is insured, and the iron casing is completely protected from the water flowing through. The wood linings are made of narrow strips, the last strip being driven firmly home, forming a key, which holds the lining in place. The whole inside is then painted with acid-proof paint, and all joints are carefully calked and painted to prevent leakage. The exterior of all pump columns is protected with acid-proof paint.

LEACHING DUMPS AND TAILINGS.

It may be of interest to state, in closing, that after the tail water, from which the copper has been recovered, has been discarded, it is collected by parties, who use it for leaching the soluble copper from old mine dumps, which are being operated under lease. These old dumps contain small amounts of copper, which was originally in the form of sulphide; but owing to years of exposure to weathering, has become more or less oxidized to sulphate.

The tail water from the precipitating plants, containing considerable ferric sulphate, is quite suitable for leaching this material. Thus some of the copper still in the form of sulphide, in addition to all the copper sulphate, can be recovered, owing to the solvent action of the ferric sulphate upon sulphide and other copper minerals.

The method of procedure is to convey the water to the top of the dump, where it is confined in shallow ponds, thus causing it to percolate through the dump. A trench is dug around the bottom of the dump, from which tunnels are run into the interior. The percolating water, collected by these tunnels, is conveyed to the trench,

from the sump of which it is pumped by electrically driven centrif-
ugal pumps to the top of a precipitating system similar to those
described above.   Water containing as high as 0.015 per cent. of
copper is obtained in this way, and, while not as satisfactory as the
mine water, is profitable to work.

<div align="center">DISCUSSION.</div>

J. W. RICHARDS, South Bethlehem, Pa. :—In looking at these pre-
cipitating launders the idea has come to me that the iron could prob-
ably be kept much cleaner if there were laid on the bottom of the
launders some sheet copper, with which the iron would be in contact,
and as the solution passed over the tendency would be to precipitate
the copper on the sheet copper and thus keep the iron cleaner
than it is under present conditions.   As soon as a piece of iron be-
comes coated with copper its surface becomes inactive, and the pres-
ence of a sheet of copper in electrical contact with the iron would
tend to precipitate the copper on the sheet copper and to keep the
iron clean.

WILLIS T. BURNS, Great Falls, Mont. (communication to the Secre-
tary*):—In connection with Mr. Febles's interesting paper a brief ac-
count of an attempt made at the Great Falls electrolytic plant to
recover the copper from the mine water by direct electrolysis may be
of interest.

A barrel of mine water of the following composition was obtained
from the Leonard mine, at Butte, Mont.

<div align="center">

Grams per Liter.

Free sulphuric acid,	0.1200
Arsenic,	0.00915
Antimony,	0.00570
Copper,	0.1573
Iron, total,	0.2310

</div>

The insoluble-anode tanks in use at the Great Falls electrolytic
plant are 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. deep inside.
These tanks contain 22 lead anodes and 21 copper cathodes, and are
operated at a current density of approximately 40 amperes per square
foot, with a drop in electromotive force of 2.4 volts per tank.

Not having sufficient mine water to permit the use of tanks of this
size, a small tank, 14 in. long, of the same proportions as the regular
tanks, was used.   The mine water was treated in this tank at a cur-
rent density of 40 amperes per square foot of cathode surface while
circulating the solution at a speed of 1 gal. per 32 ft. of cathode sur-

face per minute, equivalent to 8 gal. per minute in a standard size tank operating at 40 amperes per square foot.

While operating a tank under these conditions it was found that 44 volts per tank were required.

The following figures show the copper contents of the water before and after treatment:

	Copper Grams per Liter.
Entering tank, ...	0.1573
Leaving tank, ...	0.0629
Grams per liter removed, ...	0.0944
Per cent. extraction, ...	60.

The material deposited was a fine black slime containing 34 per cent. of copper.

If the water were treated in a standard sized tank at 40 amperes per square foot while circulating at the rate of 8 gal. per minute the power consumption would be 10,000 amperes at 44 volts, or 444 kilowatts, per tank, with an extraction of 60 per cent. of the copper content of the water.

Mr. Febles states that 37,000,000 gal. of water per month are pumped from the Leonard mine alone. Calling this figure 850 gal. per minute, $850 \div 8 = 106$ electrolytic tanks would be required to treat this water, with a recovery of but 60 per cent. of the copper. The consumption of power would be $444 \times 106 = 4,700$ kilowatts, or 63,000 h-p. consumed in the recovery of 60 per cent. of 68,000 lb. of copper per month.

The high resistance of the water and the excessive voltage required for electrolysis are due to the low free acid and copper sulphate contents of the solution. Because of the large volume of the water to be treated, the addition of a sufficient quantity of sulphuric or other acid to bring the resistance within commercial limits is out of the question. To raise the free sulphuric acid in the water to 1 per cent. would involve an expenditure of over $30,000 per month.

The cost of power per ton of copper recovered could be reduced by lowering the current density and increasing the number of electrolytic tanks. The percentage of recovery of copper could be increased by decreasing the speed of flow of the water and employing a larger number of tanks; but as the excessive voltage required made the process appear impossible from a commercial standpoint no further experiments were made along these lines.

Conclusions.—The recovery of the copper from the mine water by direct electrolysis does not appear to be possible from a commercial standpoint.

If some method of economically improving the electrical conductivity of the mine water were available the problems of electrolysis would be more attractive.

C. S. HERZIG, London, England (communication to the Secretary *):
—On p. 178 the author states:
"It has been stated that when pumping first began this water contained as much as 0.75 per cent. of copper. The statement cannot be verified, however, at this time."

In the spring of 1896 I was employed on the Anaconda mine in the machine shop, having previously been one of the assayers at the Anaconda smelter.

Some tests were being carried out on composition metal with a view to securing a mixture sufficiently resistant to warrant its use for column pipe and pump fittings.

In connection with this work I made a number of analyses of the mine water, the following being the results as copied from a note book I used at the time:

Anaconda Mine, Butte, Mont., Mar. 26, 1896.

No.	Location.	Per Cent. Cu.	Per Cent. H_2SO_4	Per Cent. Fe.
1	1,100	0.93	0.19	0.065
2	High Ore	0.05	0.30	0.061
3	1,000 east	0.92	0.15	0.092
4	1,200	0.42	0.11	0.068
5	1,200 east	1.00	0.17	0.068
6	1,200 west	1.04	0.23	1.054

From this it will be seen that as much as 1 per cent. of copper was present in the mine water. The acid determination is free sulphuric acid.

The samples in each case, if my recollection serves after this great lapse of time, were taken from the sumps and the ditches underground.

There is a considerable variation between sample No. 4 and the other two samples taken in the 1,200 level of the Anaconda mine, but my notes do not give any reason for this discrepancy. The High Ore sample is notable as containing a very much smaller amount of copper than the Anaconda mine waters, but it is interesting to note that this sample contained the greatest amount of free acid.

It may be interesting in this connection, as showing the corrosive action of this water, to give the result of a test. My note book is dated Mar. 27, 1896.

Two samples of Delta metal, marked respectively A67 and A68, were immersed Mar. 29 and withdrawn Apr. 4. The tensile strength of A67 was 60,000 lb. and the sample weighed 1 lb. 12 oz. The tensile strength of A68 was 65,000 lb. and the sample weighed 1 lb. 9 oz. The loss in weight in each case was about the same, practically 3 oz. The pieces of metal had a cross-section of about 2 by $\frac{2}{3}$ in. The loss of weight in the six days rendered its use for dump material out of the question.

At the same time a piece of aluminum bronze pipe was immersed, and lost nearly 5 oz. in four days, the weight being $14\frac{1}{4}$ oz. when immersed on Mar. 31 and $9\frac{1}{2}$ oz. when withdrawn on Apr. 4.

Notes on the Electrolytic Refining of Copper Precipitate Anodes.

BY W. T. BURNS, GREAT FALLS, MONT.

ATTEMPTS were made in 1908, at the Great Falls Works, to produce ingots direct from the Butte precipitate by smelting the material in a reverberatory refining furnace. The ingots produced in this manner averaged 99.4 per cent. of copper and 0.40 per cent. of arsenic and antimony.

The copper content being considered too low, it was decided to cast the copper into anodes and to treat the anodes in the electrolytic plant.

No. 1 Furnace, in which the precipitate was treated, was a regular cathode refining furnace previously employed in making wire bar, cake, and ingot from electrolytic cathodes. The hearth of the furnace was 24 by 12 ft., having a capacity of 100,000 lb. of cathodes. The furnace was coal fired, and the copper was dipped by means of trolley ladles.

The average composition of the furnace charges was as follows:

	Per Cent.	Per Cent. Copper.
Precipitate, ...	75.3	40 to 80
Purification cathodes, ...	1.9	92
Refinery slag (oxide slag), . .	10.1	50 to 60
Electrolytic cathodes, · · · . .	2.1	99.9
Precipitate anode scrap, ...	3.1	99.0
Scrap copper, ...	7.5	99.2
	100.0	

The slag from this furnace amounted to 46 per cent. of the total amount charged and was of the following average composition:

	Per Cent.
Cu	16.86
SiO$_2$	34.7
FeO	18.8
Al$_2$O$_3$	13.0
CaO	9.4
MgO	2.7
S	0.47
As	0.11
Sb	0.006
Ag, oz per ton	0.18

Table I. shows the weights and assays of the anodes produced.

TABLE I.—*Anodes Produced at Furnace No. 1 from Copper Precipitates.*

Lot No.	Anodes.	Weight.	Cu.	As.	Sb.	Ag. Per Ton.
		Pounds.	Per Cent	Per Cent.	Per Cent.	Oz.
1	35	19,402	99.53	0.251	0.012	0.35
2	66	37,656	99.37	0.338	0.012	0.40
3	71	44,102	99.26	0.479	0.016	0.40
4	76	47,692	99.20	0.749	0.014	0.40
5	81	52,544	98.47	0.586	0.027	0.40
6	102	59,290	98.85	0.600	0.056	0.45
7	115	70,814	98.62	0.589	0.019	0.50
8	177	105,091	99.60	0.325	0.018	0.48
9	142	84,790	98.92	0.368	0.022	0.45
1	177	95,290	98.98	0.544	0.025	0.20
2	154	88,751	99.13	0.284	0.044	0.25
3	317	171,469	99.38	0.240	0.027	0.45
4	353	206,863	98.29	0.794	0.024	0.25
5	246	150,299	98.66	0.720	0.025	0.35
6	165	94,390	99.56	0.253	0.019	0.20
7	667	366,710	99 45	0.449	0.024	0.25
8	428	231,896	99.14	0.510	0.029	0.25
9	450	240,103	99.12	0.580	0.037	0.25
10	215	113,342	99.33	0.500	0.019	0.60
11	467	237,848	99 04	0 670	0.041	0.60
12	46	23,228	99.20	0.860	0.048	0.65
13	64	36,699	99.25	0.540	0.032	0.65
14	75	44,949	99.24	0.350	0.022	0.55
15	70	42,252	99.34	0.430	0.025	0.50
16	89	53,892	99.48	0.310	0.021	0.50
Totals, 25	4,848	2,719,362	. . . . .	. . . . .	. . . . .	. . . .
Arithmetical averages, . . .	. . . . . .	. . . . . .	99.14	0.493	0.024	0.41

These anodes were refined in 32 of the regular electrolytic refining tanks. In order that a comparison between cathodes produced from these anodes at high current density and at low current density might be obtained, 4 of the 32 tanks were operated at one-half the ampere density of the remaining 28. No purification of the electrolyte was attempted until the arsenic in solution had reached 55 g. per liter, a figure 44 per cent. higher than the copper in solution. This was done so that the action of the arsenic in solution on the copper deposited might be observed. Table II. shows the records of these tanks for the six months that they were treating these anodes. It will be noted that the arsenic and antimony content of the cathodes produced in the low-density tanks averaged but 0.0031 per cent., notwithstanding the exceedingly high arsenic content of both the anodes and the electrolyte.

The cathodes produced in the high-density tanks, while higher in arsenic and antimony than the low-density product, were of a better grade than was expected.

TABLE II.—Cathodes Produced from Copper Precipitate Anodes.

At Low Current Density.

Average current density, 17.2 amperes per square foot. Average pounds copper per kilowatt-hour=7.24.

Date Drawn.	Average Amperes for Age of Cu.	Gross Weight of Cathodes Drawn.	Weight of Starting Sheets.	Net Weight Deposited.	Age Cathodes.	Tanks Drawn.	Average Ampere Efficiency.	Cathodes. As and Sb.	Cathodes. Ag.	Sp. Gr.	Acid.	Solution. Grams per Liter. Cu	As	Sb	Fe	Cl
		Pounds.	Pounds.	Pounds.	Days.	No.	Per Cent.	Per Cent.	Ounces.							
January . .	3,277	21,485	1,195	20,290	9	12	90.1	0.0024	0.01	1.231	146	41.8	20.73	0.41	5.85	0.0470
February .	4,379	33,954	1,736	32,218	8	16	87.5	0.0028	0.21	1.245	148	44.4	29.63	0.49	5.40	0.0419
March . .	4,642	33,884	1,882	32,002	8	16	86.7	0.0036	0.14	1.253	153	39.1	47.32	0.41	6.50	0.0393
April . . .	4,708	26,958	1,429	25,529	8	12	91.0	0.0033	0.22	1.260	151	38.3	55.09	0.35	6.65	0.0515
May . . .	4,700	35,514	1,985	33,529	8	16	89.8	0.0034	0.26	1.240	153	41.4	43.69	0.45	5.72	0.0490
Averages .	4,341	. .	. .	. .	8 & 9	. .	89.04	0.0031	a0.17	1.246	151	41.0	39.29	0.42	6.02	0.0437
Totals . .	. .	151,795	8,227	143,568	. .	72	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .

a Antimony constant at about 0.0015 per cent,

At High Current Density.

Average current density, 33.8 amperes per square foot. Average pounds copper per kilowatt-hour = 3.64.

Date Drawn.	Average Amperes for Age of Cu.	Gross Weight of Cathodes Drawn.	Weight of Starting Sheets.	Net Weight Deposited.	Age Cathodes.	Tanks Drawn.	Average Ampere Efficiency.	Cathodes. As and Sb.	Cathodes. Ag.	Sp. Gr.	Acid.	Solution. Grams per Liter. Cu	As	Sb	Fe	Cl
December .	8,240	231,899	12,416	219,483	4 & 5	108	84.1	0.0036	0.30	1.219	145	44.0	8.74	0.32	5.75	0.0338
January . .	6,882	333,327	17,532	315,795	4 to 10	164	86.0	0.0043	0.12	1.230	146	41.8	20.73	0.41	5.85	0.0470
February .	8,049	380,754	19,972	360,782	4 to 6	184	86.2	0.0045	0.10	1.245	148	44.4	29.63	0.49	5.40	0.0419
March . .	9,261	446,615	25,159	421,456	4	216	84.8	0.0074	0.10	1.233	153	39.1	47.32	0.41	6.50	0.0393
April . . .	9,420	438,800	25,992	412,808	4	212	83.0	0.0091	0.23	1.260	151	38.3	55.09	0.35	6.65	0.0515
May . . .	9,350	460,024	25,353	434,671	4	212	86.3	0.0070	0.24	1.240	153	41.1	43.69	0.45	5.72	0.0490
Averages .	8,533	. .	. .	. .	4 to 10	. .	85.07	0.0060	b0.18	1.241	149	41.4	34.20	0.39	5.98	0.0438
Totals . .	. .	2,291,419	126,424	2,164,995	. .	1,096	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .

b Antimony constant at about 0.0015 per cent.

The anode scrap amounted to 6.3 per cent. of the anodes treated. This was returned to the precipitate furnace.

Table III. shows the conductivity and the arsenic and antimony content of the furnace products obtained from the treatment of about one-half of the cathodes; the remainder of the cathodes were mixed with other products for furnace treatment.

TABLE III.—*High-Density Cathodes from Copper Precipitate Treated at Furnace No. 2.*

Cathode Dock Lot No.	Weight of Cathodes	As and Sb.		Conductivity		Dipped Into.
		Cathode Sample.	Furnace Sample	Hard Drawn.	Annealed	
	Pounds.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
1	89,354	0.0031	0.0041	96.90	99.50	Plates and ingot bar.
2	88,686	0.0037	0.0043	97.70	100.20	Ingot bar.
3	89,021	0.0043	0.0036	97.50	100.20	Ingot bar, B.M. ing.
4	88,144	0.0031	0.0033	97.30	100.00	Ingot bar.
5	92,193	0.0046	0.0044	97.30	99.90	Cake.
6	84,793	0.0053	0.0053	96.10	98.70	Ingot bar, M.A. ing.
7	92,112	0.0050	0.0031	96.60	99.30	Cake, ingot bar.
8	94,912	0.0041	0.0034	96.50	99.20	Cake, ingot bar.
9	87,175	0.0046	0.0041	97.40	100.20	Cake, ingot bar.
10	90,081	0.0043	0.0052	93.50	96.10	Ingot bar, M.A. ing.
11	83,467	0.0043	0.0072	95.50	98.20	Ingot bar, M.A. ing.
12	93,291	0.0053	0.0058	94.40	97.10	Ingot bar.
Totals, 12	1,073,229	. . . . .	. . . . . .	. . . .	. . . . .	. . . .
Averages,.		0.0043	0.0045	96.39	99.05	. . . .

NOTE: The "cathode sample" was obtained by punching three 0.25-in. holes in every tenth cathode.

The "furnace sample" is the regular granulated furnace sample and should be more reliable than the cathode sample.

The electrolytic slime obtained from the treatment of the precipitate anodes was very voluminous and much remained in suspension in the electrolyte. The following analysis is of a sample of dried slime:

	Per Cent.
Cu,	65.22
S,	17.20
As,	3.40
Sb,	2.22
Sn,	2.24a
Pb,	1.47
Fe,	1.00
Insol.,	1.00
Ag., oz. per ton,	100.90
Au, oz. per ton,	0.56

a Probably from cans used in the precipitating process.

The slime produced was 31,948 lb., equivalent to 23.5 lb. of slime per ton of anodes, as compared with 18 lb. of slime per ton of regular converter anodes. This slime was treated in the reverberatory matte furnaces.

Conclusions.

Cathodes of wire-bar grade can be economically produced from copper precipitate anodes when treated at a current density of 17 to 18 amperes per square foot.

Cathodes, from which cake and ingots averaging 99.85 per cent. of copper can be made, can be economically produced from copper precipitate anodes at a current density of 33 to 35 amperes per square foot.

The removal of the arsenic from the electrolyte is the principal item of electrolytic refining cost that is in excess of the cost to refine ordinary converter anodes.

It would then appear that if anodes can be produced as cheaply in the refining furnace as the precipitate can be treated in the smelter, there is little choice between the two methods, as the copper must eventually pass through the electrolytic refinery.

Mining Cost Accounts of the Anaconda Copper Mining Co.

BY H. T. VAN ELLS, NEW YORK, N. Y.

THE following is a brief description of the cost accounts in effect at the mines of the Anaconda Copper Mining Co.

The accompanying chart, Table I., shows the distribution of labor, materials, and redistributable accounts to the various operating and repair accounts.

OPERATING.

Development.

Owing to the fact that every mine has its peculiarities in formation and methods, the question of arriving at what charges should be included under the head of "Development" is one that has occasioned considerable thought and comment, and in the cost sheet of the Anaconda Copper Mining Co. this account has been sub-divided to show the expenses of breaking ground, tramming, timbering, shaft sinking, up-raising, and cutting stations and skip chutes. It has always been the policy of the company to charge the entire cost of this work to operations monthly, whether the mines are producing or not, also the cost of the opening up of new mines; and only surface construction and additions are charged to capital, excepting original machinery installed for operating pumping plants. Explanations of the charges to the various sub-accounts follow :

Breaking Ground.—Work performed in driving cross-cuts from the station, or cross-cuts from the drifts, and diamond-drill work, for the purpose of discovering or developing ore bodies.

Tramming.—Cost of tramming waste produced by cross-cuts mentioned above. When waste is hoisted to another level to be used for filling, the charge to this account stops at the station, and when hoisted to surface the charge includes tramming from the collar of the shaft to the dump.

Timbering.—Cost of timbering the cross-cuts mentioned above includes the cost of handling and framing such timbers, but does not include the wages of station tenders and top carmen.

No repairs and renewals are included in above costs, as they are charged to repair accounts.   This is true of all the following operating accounts, except as otherwise stated.

Sinking and Up-raising.—Charges under this heading are classified as follows:

Breaking Ground.—Includes all charges for sinking and up-raising, when shaft timber is used.

Tramming.—Cost of tramming waste.   When waste is hoisted to surface, in addition to the wages of the shovelers there is included the cost of tramming the waste material from the collar of the shaft to the dump, and when hoisted to another level only the wages of the shovelers is included.

Timbering.—Cost of timbering shafts and up-raises, expenses of handling and framing timbers, and cost of compressed air used in hoisting timbers when up-raising.

Draining.—Cost of draining or unwatering shaft, including repairs to suction hose and repairs to and handling of sinking pumps; also cost of power used in operating pumps.

Hoisting and Lowering.—This includes expenses of operating auxiliary hoist while sinking shaft, repairs to sinking cages, buckets, shaft-sinking engine, etc., and power used by engine, not including repairs to main hoisting equipment.

The statistics shown under this heading are: number of feet advanced and cost per foot, and cubic feet excavated and cost per cubic foot.

Cutting Stations and Skip Chutes.—Charges under this heading are classified as follows:

Breaking Ground.—Cost of breaking ground for stations, and cost of excavating and putting in skip chutes, except where a station is cut for pumps and used exclusively for that purpose.

Tramming.—Cost of tramming waste and putting in new turnsheets and repairs to same while cutting station.

Timbering.—Cost of timbering stations and skip chutes and expenses of handling and framing such timbers.

Hoisting and Lowering.—Cost of hoisting and lowering, repairs to sinking cages, buckets, shaft-sinking engine, etc., and cost of power used by engine.

A separate statement is made for each station or skip chute cut, and under Statistics is shown the number of cubic feet excavated and cost per cubic foot.

Draining.

The cost of draining or unwatering mines includes wages of miners

engaged in digging ditches, putting in water boxes, pumpmen, cost of new drain tunnels, cost of pumping water into drain tunnels, proportionate part of draining by pumping plants based on the gallons of water pumped per minute, cost of diamond-drill work for drainage purposes, and cost of power used in operating pumps.

Mining.

Breaking Ground.—Work performed in drifts, up-raises, winzes and stopes, lateral drifts and cross-cuts from laterals to main drift, cross-cuts run for the purpose of obtaining filling, breaking and shoveling waste on surface when used for underground filling, cost of putting in doors for ventilation, and cost of compressed air used for ventilation and drills.

Tramming.—Cost of tramming from drifts, up-raises, winzes, and stopes to the shaft, and on surface from shaft to ore bins, tramming waste on surface for underground filling, and on levels the cost of tramming from shaft to waste chutes; cost of shoeing horses and mules, new track and turnsheets in drifts, lateral drifts and lateral cross-cuts, and cost of electric power used in operating motors.

Timbering.—Cost of timbering in drifts, up-raises, winzes, and stopes, lateral drifts, lateral cross-cuts, new ore chutes, raising of chutes from floor to floor, cost of handling and framing timbers used, cost of operating timber hoists for hoisting timber into the stopes, and cost of power used by same.

Hoisting and Lowering.

Wages of engineers on main hoist, auxiliary hoist, underground engines, oilers and wipers, station tenders engaged in handling ore, waste, materials and supplies, hoisting charges when ore is hoisted through another shaft, cost of compressed air and steam power used by hoisting engines, and cost of steam furnished for heating air in receivers.

Superintendence.

Salaries of the Superintendent, shift bosses, foremen, and clerks, including residence repairs and maintenance of horses and vehicles used by these employees.

Fire Expense.

This being an extraordinary expense it is considered necessary to show it under a separate account. It includes all charges for controlling and fighting fire underground.

Miscellaneous.

Wages of timekeepers and watchmen, and charges from redistributable accounts which cannot be properly charged to other operating accounts, such as assaying, lighting, teaming, change house, surface, shop maintenance, and precipitating underground.

REPAIRS AND RENEWALS.

Hoisting Equipment.—Cost of repairing or renewing air receivers, air pipes from receivers to engine, engines, cables, head frame, cages, skips, bell ropes, signal lines, etc.

Shafts and Stations.—Cost of repairing, renewing or retimbering shafts, stations, and skip chutes, and repairs and replacements of turnsheets in the stations.

Levels.—Cost of repairing or renewing underground timbering, except such repairs as are otherwise stated.

Draining Equipment.—Cost of repairing drain tunnels, pumps and accessories, pipes, water boxes, tanks, etc.

Cars, Motors, and Track.—Cost of repairing or renewing cars, motors, tracks, turnsheets, switches, etc., whether on surface or underground.

Timber Hoists.—Cost of repairing or renewing timber hoists, trucks, etc.

Air Drills and Hose.—Expenses of repairing or renewing air drills, hose, tripods, bars, etc., and replacing parts of same, which includes all repairs from the connection at the air pipe, except sharpening or repairs to steel for drills.

Hammer Drills and Hose.—(Same as Air Drills and Hose.)

Tools and Utensils.—Cost of repairing, renewing, or adding to tools and utensils used in operating the mines, including wages of toolmen engaged in carrying tools, and wages of tool sharpeners when engaged in sharpening steel for drills, picks, tools, etc.

Ore Bins.—Cost of repairing, renewing, or adding to shaft ore bins and railroad ore bins, including foundations, trestles, approaches and all permanent attachments thereto.

General.—Cost of making repairs or renewals not otherwise provided for.

GENERAL EXPENSES.

Items making up this expense, and which are self-explanatory, are as follows:

General office, legal, traffic, telephone, engineering, geological, donations, taxes, and insurance. The distribution to the various mines is made to (1) mines direct where such charges are obtainable, and (2) balance proportioned on tonnage basis.

Redistributable Accounts.

Power House.—Wages of firemen, coal passers, and ashmen, and all work performed for the purpose of developing steam power; also all charges for repairing or altering buildings, ash flumes, ash houses, water tanks, boiler feed pumps, repairing or cleaning boilers, and premiums paid on boiler insurance. Distribution is made as " Steam Power " to the various operations on the basis of horse-power furnished.

Compressed Air.—Wages of compressormen, oilers, and wipers, etc., tending compressors and motors, pipemen and others when engaged in or repairing air pipe lines; cost of repairing buildings, oil houses, compressors, and electric motors used in operating compressors; cost of steam and electric power used for operating compressors. Distribution is made to the various operations on the basis of horse-power furnished.

Pumping Plant.—Wages of pumpmen running pumps and all work performed in pumping water from the drain tunnel to surface through tunnels to precipitating boxes; also cost of repairing or renewing pumps and accessories, pump stations, etc.; cost of steam and electric power used for operating pumps. Distribution is made to " Draining " expenses of the different mines on the basis of gallons of water pumped per minute.

Change House.—Wages of men employed in taking care of change house, also all work performed in making repairs or alterations thereto, steam furnished for heating, electric current for lighting. Distribution is made to " Miscellaneous " expenses of the different mines on the basis of total number of men employed.

Shop Maintenance.—Wages of men employed in making or repairing any kind of tools, steam hammers, or machinery used in shops or connected therewith; also all labor performed in repairing or altering shop building. Distribution is made to " Miscellaneous " expenses. Work performed by shopmen either underground or on surface is distributed daily to the proper repair account.

Electric Light.—Wages of electricians and helpers employed in the repair and maintenance of light lines and lamps on surface or underground, electric current furnished for lighting, cost of all supplies, such as lamps, shades, globes, carbons, wire, and electrical fixtures. Distribution is made to " Miscellaneous " expenses of the different mines on the basis of number of lights used.

Electric Plant.—Wages of men employed in tending motors, generators, switchboards, transformers, etc., cost of repairing or renewing machinery, cost of current purchased. Distribution is made

as " Electric Power " to the various operations on the basis of power used.

Assaying.—Wages of assayers, chemists, sample grinders, and helpers employed in assay office; also work performed in repairing or renewing buildings and machinery. Distribution is made to " Miscellaneous " expenses of the different mines on the basis of number of determinations made.

Surface.—Wages of men employed in grading and repairing roads, cleaning up yards, etc., and all surface work not chargeable to other specified accounts. Distribution is made to " Miscellaneous " expenses of the different mines on the basis of work performed.

Teaming.—Wages of teamsters and stablemen engaged in general teaming, caring for horses and mules, repairing stables, vehicles, and harness. Distribution is made to the various accounts on the basis of work performed.

Precipitating.—Wages of men employed in precipitating copper from mine water underground, including building of boxes, ditches, etc., is distributed to " Miscellaneous " expenses of the mine producing such precipitates. Surface precipitating plants are not distributed to expenses of any of the mines, as the expense of operating such plants could not be properly so charged; these expenses are kept separate.

Miscellaneous.—Sales of power, air, and light to outside mines and parties are credited to above expenses, and the net cost distributed to power, air, and light of the various operations. The amount realized from sale of scrap iron to outside mines and parties is credited to Sundry supplies under " Miscellaneous " account.

Deductions from " Operating " expenses are made for labor performed and supplies furnished outside mines and parties which have been charged to operating, but no deductions are made for revenues from rents, such as buildings, machinery, shafts, or dwellings, and royalties from ore and precipitates.

GENERAL REMARKS.

After the costs as enumerated above have been shown on the cost sheet, the following information is given under the head of " Statistics ": Wet tons of ore extracted, per cent. moisture, dry tons extracted, per cent. copper and contents, ounces silver per ton and contents. Also an efficiency statement showing (1) number of days in operation, (2) average number of shifts per day, (3) average number of tons per day, and (4) average number of tons per shift per day.

On account of the number of mines operated by the company some system had to be perfected in order to make a concise and intelligent

cost sheet, and it is believed that with the introduction of the present system this has been accomplished, consisting as it does of only two sheets for each mine, without details of redistributable accounts, which adds approximately two sheets.

The cost sheet as finally prepared gives the total cost of the above accounts, with the cost per ton; classification of labor, divided between operating, shops, and superintendence, with the shifts worked, total cost and cost per ton; and classification of supplies, with the total cost and cost per ton.

All labor distributions underground are made by the foremen, turned over to the timekeepers, completed, and sent to the mine office for tabulation and verification. Each day's payroll is checked with the distribution to the various accounts, which saves a great deal of time at the end of the month. When the last day's payroll and distributions have been checked the monthly payroll and distributions are complete, usually about the second of the following month.

Statements are made monthly showing the extraordinary expenses, if any, which have been included in the operating or repair accounts. Under this head may be classified such items as expenses of mines during shut-down, special repairs to hoisting engine, gallows frame, retimbering of shafts, etc. This information in the past has been of great value in making comparisons on the cost of extracting ore as between periods.

The Statistical department each period prepares a report showing the detailed cost of mining as compared with the former period, and such items as the above are taken into consideration, as is also increase or decrease in costs of development and shaft sinking, increase of wages, fire expense, etc., and the final results show whether ore has been extracted at a lower or higher cost and the reasons therefor.

In the Statistical department all the costs per ton shown are tabulated monthly, and a report is furnished condensing the cost sheet to such an extent that costs per ton of the various items at each mine are seen at a glance, and remarks made covering the reasons for any irregularities.

A monthly statement is prepared showing the cost of production per pound of copper from each of the mines operated, carrying the ores treated through each one of the processes at the smelter, arriving at a result for the treatment and production from such ores. This statement has been made for several years, and such efficiency has been attained that monthly figures check the final results for the year when balance sheets are prepared.

TABLE I.—*Chart Showing Distribution of Labor, Material, and Redistributable Accounts.*

	Direct Charges		Redistributable Accounts											
	Labor.	Materials.	Steam Power	Compress'd Air.	Pumping.	Change House	Shop Maintained.	Electric Light.	Electric Power.	Assaying	Surface	Teaming.	Precipitat.	General Expenses.
OPERATING.														
Development:														
Breaking ground	—	—	..	—	.	.		.	.		—		..	.....
Tramming	—	—	..					.	.		...	—	.	....
Timbering	—	—	.				..	.		.	.		. .	....
Sinking and up-raising	—	—		—			. .	..		.		.	..	...
Cutting station	—	—		—		.		..		.		.. ...	.....	...
Draining	—	.		—	—	. .			—			..	.. ....	
Mining.												.. .		
Breaking ground	—	.		—	. .		.	.	—			—		....
Tramming	—	.			. ..		.	.	—			—	.	....
Timbering	—	.		—		. ..		.	—		. .	.	.	
Hoisting and lowering	—	—	. .	—		.	.	.	—		. .	.	...	.
Superintendence	—	—	. .	.		. .	.	.				. .		.. ..
Fire expense	—	—	. .		—	. .		.	. .			.	.	.
Miscellaneous	—	—	. .		. .	—	—	—		—	—	—	—	.
Total	—	—		—	—	—	—	—	—		—	—	—	....
REPAIRS AND RENEWALS.														
Hoisting equipment	—	—	..	.....	. .	..	. .	. .	. .	...		.	. ..	
Draining equipment	—	—			. .	. .	.	.	. .		..		.. ..	..... .
Shafts and stations	—	—		—	. .	.	. .	.	. .	..	..		... ..	.....
Levels	—	—			. .	.	. .		..	...		... ..	.. .
Cars, motors, and track	—	—	....		. .	....		. . .	. .	....	.	.....	..
Timber hoists	—	—		. .	.....	..	.. .	. .	. .	.....	. . .	.. .	..
Air drills and hose	—	—	....	. ...	.. .	..	. .	..	.....	.	.. .	.	..
Hammer drills and hose	—	—	. ...	. .....	.. .	.	. .	..	. . .	.	. .	...	.....
Tools and utensils	—	—		. .	. ..	. .	.	...	.....		..		.. ..
Ore bins	—	—	. .	...	....	..	.	.	...		...	..
General	—	—	. .	..	. .	.....		..	...	..	.....		...
Total	—	—	..	..	...	.....	..	.	.	... .	.
GENERAL EXPENSES.														
Charged direct	—	—	. .		...	.. .	. ..		.. .	..	. .		.....	...
Proportioned	..	... .		. .		.	....	...	....	...	.	.	. .
Total	—	—	.	.		.	.. ...	....	.		. .	. .	—
Grand Total	—	—	—	—	—	—	—	—	—	—	—	—	—	—

The Great Falls System of Concentration.

Installed in Section No. 1 of the Washoe Concentrator at Anaconda, Mont.

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THE copper-bearing sulphide ores from the mines in Butte, Mont., which are for the most part concentrated at the Boston & Montana Reduction Works in Great Falls and at the Washoe Reduction Works in Anaconda, contain the following minerals of economic value: Chalcocite, Cu_2S; enargite, Cu_3AsS_4; cupriferous pyrite; bornite, Cu_3FeS_3; covellite, CuS; tetrahedrite, $4 CuS_2Sb_2S_3$; and tennantite, Cu_8AsS_7. All of the ores carry a high percentage of pyrite. Many of the above sulphides are argentiferous and also carry from 10 to 20 cts. per ton in gold. The gangue is quartz and highly altered granite, with sphalerite and galena and more rarely barite and hubnerite as accessory minerals.

An average chemical analysis of the second-class ore concentrated at Great Falls and Anaconda would be as follows:

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Copper  .. ... .. . . . ...... .. ..... .3 2 to 3.4 per cent.
Silver.. . .... .. .. . . ... .. ... ..2.6 to 3 0  oz. per ton.
Gold.... . ...... .. .. . . . . .......0.07 to 0 15 oz. per ton.
SiO₂..... ............... ..... . ....... 55 to 60 per cent.
FeO..... ........... ........ . 12 to 15 per cent.
Sulphur....... . .................. ........ . 12 to 15 per cent.
Al₂O₃.......... . ... ............. .. 8 to 10 per cent.
CaO........... ................ . .......0.5 to 0.8 per cent.
As₂O₃......... ................. ...........0.5 to 0.6 per cent.
Lead........ . ... ...... .... . .......0.2 to 0 3 per cent.
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Before describing the Great Falls flow sheet as installed at Anaconda, a brief history of the Great Falls concentrator will be given, including a description of the various concentrating machines and systems which have been tested in the search for a flow sheet which would give a higher recovery of copper.

I. EARLY HISTORY OF THE BOSTON & MONTANA MILL AT GREAT FALLS.

a. Original mill.—This mill was erected during 1891 and began to treat the ore from the Boston & Montana Co.'s mines in February, 1892. As first built, the mill consisted of three sections, rated at 300 tons each. The third section was not put into operation until about a year after the first two began to operate. At this time there were no blast furnaces at the Great Falls plant, all of the concentrate being roasted and then smelted in tilting furnaces. On this account the concentration did not begin until all the ore had been crushed through 0.5 in. Blake crushers and rolls were used for the coarse crushing, and the first jigging material was the undersize of 0.5 in. and the oversize of 8 mm. This product was treated on two single-sieve slide-arm jigs per section, manufactured by the Fraser & Chalmers Co. These jigs made a concentrate and a middling, the latter product being crushed through rolls to 8 mm. The total 8 mm. undersize was sent to V classifiers, the spigots of these classifiers going to

three rows of Collom jigs and the overflow to V tanks feeding the round tables. The jigs made a concentrate, a middling, and a tailing. The middling was crushed in rolls to about 2.5 mm. and fed to a second classifier, the spigot of this classifier going to a fourth row of Collom jigs and the overflow to the round-table feed tanks. These jigs made only concentrate and tailing. The round tables were wooden-deck convex buddles about 18 ft. in diameter. These tables made a concentrate and a middling but no tailing. The middling was treated on 4-ft. Frue vanners, making a concentrate and a tailing.

b. Coarser concentration.—About 1894, blast furnaces were erected at the Great Falls plant. In order to obtain coarser concentrate for these furnaces, the one-sieve slide-arm jigs were replaced by two-compartment Harz jigs and the limiting size of crushing was increased from 0.5 in. to about 0.75 in. Shortly after this, in order to obtain more coarse concentrate, the limiting size was increased to 1.25 in. and two one-compartment Harz jigs were added to each section to treat the undersize of 1.25 in. and oversize of 0.75 in., the material through 0.75 in. and over 8 mm. going to the two-compartment Harz jigs.

c. Screen sizing of fine jig feed.—The next improvement was the substitution of screen sizing for the feed to the three rows of Collom jigs in place of the very imperfect classification. Two sets of trommels were added,—5 mm. round hole and 2.5 mm. round hole. The material between 5 and 8 mm. was sent to one set of jigs and that between 5 and 2.5 mm. to a second set. The undersize of the 2.5 mm. trommels was classified in an Evans launder classifier and sent to the third set of jigs. This classifier made four spigot products and an overflow. Each spigot was treated on a separate jig and the overflow was sent to the round-table feed tanks. About this time the Collom jigs were replaced by Harz jigs of the Evans type. The middling from these three sets of jigs was crushed through rolls as before, but a 2 mm. round-hole trommel was added to the roll system, thus forcing the rolls to crush through 2 mm.

d. Installation of finer grinding.—It was found that the tailing from the roll middling jigs was running too high in copper, and to overcome this a middling was made from these jigs, which was sent to 5-ft. Huntington mills for finer grinding. These mills were equipped with slotted screens, the openings being 1.5 by 10 mm. Soon after this a middling section was built, equipped with 5-ft. Huntington mills and Harz jigs to treat the middling from the 2.5 mm. and the roll middling jigs. The Harz jigs were fed from Evans launder classifiers, the overflow of the classifiers being returned to the table divisions of the main sections.

e. Stamp tried in place of Blake crushers and rolls.—About 1893 a steam stamp was installed in Section No. 3 to replace the Blake crushers and rolls in the coarse-crushing section. This stamp was equipped with

screens having openings of $\frac{3}{16}$ by $\frac{3}{8}$ in. Later the openings were increased to as large as $\frac{7}{8}$-in. round hole, but it was found that the stamp made altogether too much slime, and it was soon discarded.

f. Mill increased from three to six sections.—During the year 1900 three more sections were added to the mill, the equipment in the new sections being the same as that of the first three sections. A Huntington mill middling section was added to treat the fine middling from the three new sections.

g. Tonnage increased from 300 to 450 per section.—The tonnage was gradually increased from the start and by 1901 the mill was treating 2,700 tons a day or 450 tons per section.

h. Round tables replaced by Wilfley tables.—During 1901 the round tables in Section 6 were replaced by Wilfley and Overstrom tables. These tables gave very satisfactory results, producing a much cleaner concentrate than the round table and making a considerable amount of tailing, which reduced the load on the vanners following. A sufficiently clean tailing to be discarded could not be made from the round tables. The following figures taken from tests made in September, 1903, give the analyses of the concentrates from the two different tables:

	Cu Per Cent.	SiO$_2$+Al$_4$O$_3$ Per Cent	Fe. Per Cent.
Concentrate from round tables	11 55	48 1	14.1
Concentrate from Wilfley tables.. .. .	16 57	25 2	21 5

The round tables in all but one of the remaining five sections were replaced by Wilfley tables in 1905.

There is, of course, no question to-day concerning the relative merits of a round table and a reciprocating table for the treatment of sands. There is, however, a splendid field for the round table in the concentration of slime.

II. The Hancock Jig Replaces the Evans Jig.

During 1905 a 25-ft. Hancock jig was installed in one section of the mill to be tested against the Evans jig. The Hancock jig has proved to be quite superior to those of the Harz type, to which class of jigs the Evans belongs, and has been incorporated in the flow sheet as installed in the Washoe concentrator at Anaconda. For this reason the construction and operation of this jig will be described briefly and an account of some of the experimental work that was done at Great Falls in perfecting this jig will be given.

a. Description of Hancock jig.—The Hancock jig belongs to the movable-sieve type of jigs. It is of Australian origin and was invented by H. R. Hancock. The larger size jig consists of a tank, A, Fig. 1, 25 ft. long by 4 ft. wide and 5 ft. deep, in which is suspended a frame, B, 20 ft. long by 2 ft. 8 in. wide. This frame carries a series of screens, the sizes of the apertures depending upon the material to be concentrated. These screens rest on hard-wood slats spaced 5 in. between centers. Above the screen are brass castings, C, forming pockets 5 by 10 in. by 3 in. deep. The transverse ribs of these castings fall immediately above the ribs supporting the screens. The brass castings are held firmly against the screen by side boards, D, which are in turn held securely in place by hard-wood wedges, E, driven in between them and steel lugs, F, on the side boards of the tray. The tray is carried on two cast-steel cross-bars, G, which are in turn connected to the levers, H, by the connecting arms, J, on either side of the jig hutch. These levers engage a three-way cam, K, on the main drive shaft, L, of the jig. This shaft revolves at 60 to 65 rev. per min. and imparts an up-and-down motion to the tray through the rocking arms and connecting rods. The upward movement of the tray is uniform, while the downward movement is accelerated and accompanied by a "bump." The up-and-down stroke ranges from $\frac{3}{8}$ to $\frac{3}{4}$ in. in length. The connecting arms at the head end of the jig are connected to the jig hutch by adjustable quadrant arms which impart a forward-and-backward motion to the tray. Both the up-and-down and the forward-and-backward motions can be adjusted to suit the various requirements of the different pulps to be treated. Bedding material, consisting of coarser mineral particles, iron slugs, or punchings, etc., is retained in the pockets above the screen. The feed is introduced on the tray at the head end and travels the length of the tray. The concentrate is drawn down through the bedding and screen into the hutch from which it discharges, while the gangue, or lighter material, being unable to penetrate the bed, is carried along the tray and discharged over the end. The ore bed on the jig is usually about 4 to 6 in. deep above the bedding and the water level in the tank or hutch is maintained about 4 in. above the top of the ore bed.

The first jig installed at Great Falls was defective mechanically and did not stand up well. Since this time a great many improvements in the mechanical design of the jig have been made at Great Falls, so that to-day the jigs in use there give very little trouble from breakdowns and all the wearing parts are accessible and easily replaced with but little lost time.

b. Tests made on Hancock jig at Great Falls.—In this paper the term "natural feed" will be applied to a pulp which has neither been classified nor sized. For example, the undersize of a screen which is subjected to
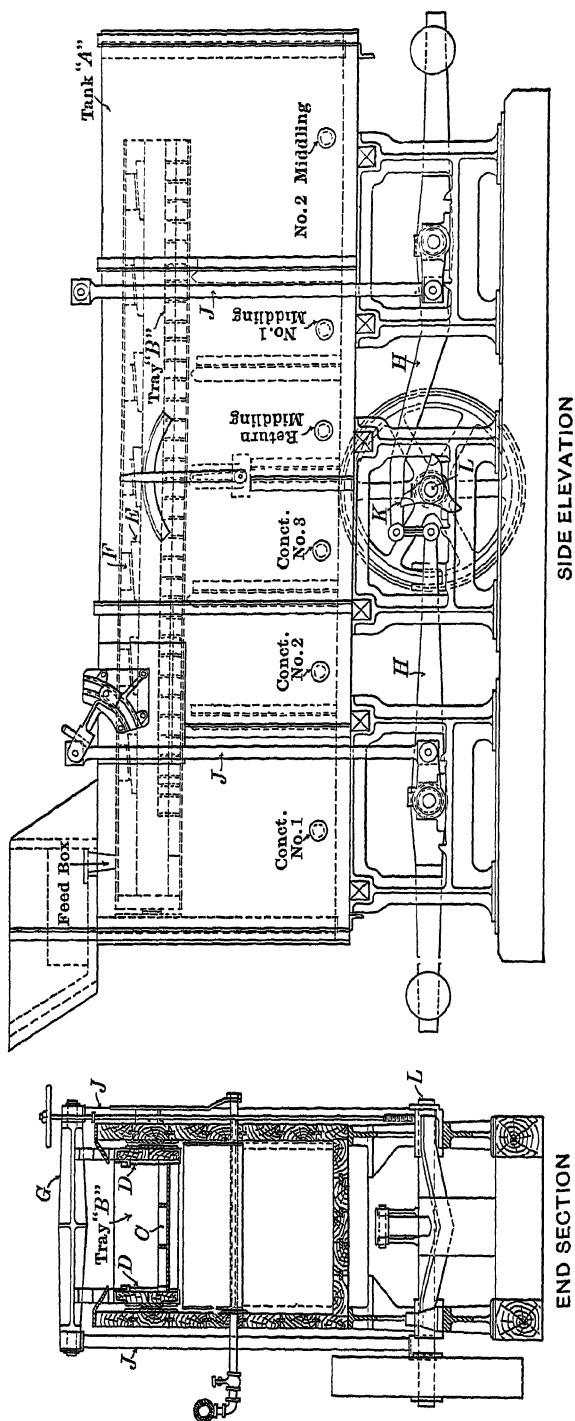
FIG. 1.—HANCOCK JIG.

no further sizing or classifying but contains particles ranging in size from the given screen size down through the finest slime, would constitute a "natural feed" for a machine. A pulp in which the range of size of the particles is limited by some sizing device will be termed a "sized feed." The product of a classifier will be termed a "classified feed." However, it should be noted that all products from a classifier or classifier system are not truly classified products, as the coarsest product from a classifying system or device will contain an increment of coarse mineral grains while the finest product will contain an increment of fine gangue material.

1. *Treating natural feed, undersize of 8 mm. round-hole trommel.*—The first tests, made in June, 1905, were upon the treatment of the undersize of the 8 mm. round-hole trommels. In the table on page 217 the results obtained from two typical tests upon this class of feed are given, one in which the jig was treating a low tonnage and one in which the jig was treating a high tonnage. The material was fed directly to the jig, with no dewatering or desliming. Referring to the tabulated results of these two tests, it will be noticed that the low-tonnage test shows a very good recovery of copper, whereas the feed treated in the high-tonnage test was evidently too heavy for the jig to handle efficiently. The consumption of water is high on this class of feed, due to the necessity of adding a considerable quantity of water beneath the tray of the jig to keep the slime and fine sand out of the concentrate. The Evans equipment replaced by the one 25-ft. Hancock jig in the low-tonnage test consisted of eight double-compartment jigs treating 8 to 5 mm. material, eight double-compartment jigs treating 5 to 2.5 mm. material, two four-spigot classifiers treating the undersize of the 2.5 mm. trommels, eight double-compartment jigs treating the spigot products of the two classifiers, two 5 mm. trommels and two 2.5 mm. trommels, the preceding being the equipment for that division of a 500-ton section. In the high-tonnage test the above equipment in two 500-ton sections was replaced by the one Hancock jig.

2. *Treating sized material, 8 to 2.5 mm. round hole.*—A test was made in March, 1909, upon a sized feed, 8 to 2.5 mm., which has been chosen as being typical of the Hancock-jig work on this class of feed. During this test the undersize of 8 mm. and oversize of 2.5 mm. from two sections were treated on one Hancock jig. One 25-ft. jig replaced 16 two-compartment Evans jigs treating 8 to 5 mm. material, 16 two-compartment Evans jigs treating 5 to 2.5 mm. material, and four 5 mm. trommels. The jig produced a better grade of concentrate from this sized feed than from the natural feed, used considerably less water, and made lower-grade middlings, although the percentage of recovery is less on account of the lower grade of feed.

3. *Treating classified material, 11 to 0.25 mm. quartz.*—A test on this

class of material made during November, 1910, has been used to demonstrate the work of the jig. During this test the jig was fed from an Anaconda classifier designed at the Boston & Montana mill. This classifier received as feed the total undersize of an 11 mm. square-hole trommel from one 500-ton section. The overflow of this classifier was sent to the Wilfley-table feed tanks and the plug discharge was treated on the jig. The preceding tests were all made on a 25-ft. jig with a 20-ft. tray, whereas in this test a 19-ft. jig with a 15-ft. tray was used. Following are screen sizing tests made on the spigot product and overflow of the Anaconda classifier preceding the Hancock jig:

Sizing Tests Made on the Spigot Product and Overflow of the Anaconda Classifier.

Hancock Jig Treating Spigot Product. Wilfley Tables Treating Overflow.

SCREEN SIZES	SPIGOT PRODUCT			OVERFLOW		
	Cumulative Per Cent. Solids	Assay Per Cent. Cu	Cumulative Per Cent. Cu.	Cumulative Per Cent. Solids	Assay Per Cent. Cu.	Cumulative Per Cent. Cu.
On 11 3 mm.. ...	0 5	3 05	0.5	.. . ..	. .. .	.. . .
On 8 0 mm......	12.3	2 38	10 1	.. . . .	. .. .	. .
On 5 7 mm	28 8	2.18	22 4	.....	. .	. .
On 4.0 mm......	42 7	2.65	35 0	....	....	....
On 2.83 mm......	52 4	2 66	43 8	.. . .	. . .	. .....
On 2 00 mm......	58 5	2 83	49 7	. ... .	. . ...	. .....
On 1.41 mm.. ...	68.9	2 88	60.2	.	. . .	.. .
On 1 00 mm......	74.5	2 98	65 9	.	. . . .	. .
On 0 71 mm......	82.7	3.07	74 5	.....	. . . ..	. . .
On 0.50 mm......	87 4	3 12	79 5	... . .	. .. .	. . ..
On 0.25 mm......	93.3	3 53	86.6	2.4	0 90	0 6
On 0 110 mm.. ...	97 4	5.86	94 8	16 1	1 36	6.4
On 0.075 mm..	98.4	6 74	97 1	31 6	3 47	23 2
Through 0.075 mm...	1 6	5 31	2 9	68.4	3.59	76 8
TOTAL.....	.. .	2 93	. . ...	. ..	3.20	.. . ...

The jig did remarkably good work upon this classified feed, producing a middling assaying 1.10 per cent. of copper from a feed assaying 2.95 per cent. of copper.

c. Tests made on Evans jigs.—During June, 1905, a test was made on the Evans jig system and the results of this test are included in the tabulation on page 217, as being typical of the work of these jigs. The system comprised eight double-compartment jigs treating 8 to 5 mm. material, eight double-compartment jigs treating 5 to 2.5 mm. material, and eight double-compartment jigs treating the classified 2.5 mm. undersize, slime removed. This system produced very clean concentrate, but made a lower recovery of copper and used a great deal more water than

the equivalent Hancock jig system. The system produced a small amount of tailings amounting to 4.7 per cent. of the original feed and assaying 0.97 per cent. of copper.

The percentage recoveries of copper given in the tabulated results on this page, as well as in some of the test data to follow, are based upon the total copper in the feed to machine or system. Our more recent practice is to base the recovery, not upon the total copper in the feed, but upon that copper which is in such a form in the feed that it may be recovered in a suitable grade of concentrate by direct mechanical concentration. This latter method gives, of course, the true efficiency of the machine or system under test.

Tests Made on Hancock Jig at Great Falls, Mont.

DATA	NATURAL FEED Through 8-mm. round-hole trommel (Slime contained)		SIZED FEED Through 8-mm. on 2.5-mm. round-hole trommel	CLASSIFIED FEED 11-mm. to 0.25-mm. quartz	TEST MADE ON EVANS JIG SYSTEM AT GREAT FALLS, June, 1905
	Low Tonnage	High Tonnage			
Feed per 24 hours:				.	
Average.	420	865	545	480	430
Maximum.... . .	450	980	. . . . .	. . . . .	480
Minimum	400	750	. . . .	. . . . .	420
Feed assay, per cent. Cu.. .	3 31	3 43	2 71	2 95	3 38
Concentrate assay, per cent. Cu..	9 00	9.15	13 90	9.58	10.5
Concentrate assay, Insol (SiO₂+Al₂O₃)	30 7	25.8	18 7	23 8	12.3
Middling assay, per cent. Cu....	1 56	2.04	1 46	1.10	1 70
Per cent. copper recovered .	59.2	49.6	51.6	58.72	48 7
Machines displaced by Hancock Jig:					
Evans jig compartments ..	32	64	64	72	....
Evans classifiers	2	4	None	3	. . . .
Trommels, 3 by 6 ft..... .	4	8	4	4	... .
Screen area of Hancock jig, sq. in................. .	6,690	6,690	6,690	5,500	. ...
Screen area of Evans jig retired, sq. in.	36,500	73,000	48,670	54,720	....
Water used per ton per 24 hours, gals.............	1,530	....	350	515	3,500

General notes on Hancock jigs.—The chief advantages of the Hancock jig over jigs of the fixed-sieve type are:

Higher recovery of copper;

Great saving in floor space;

Great saving in water consumption;

Less operating and repair labor;

Less power required.

We have not found it advantageous to produce a tailing from the Hancock jig, the usual practice being to make concentrate from the first

three hutches, a middling from the fourth hutch, which is returned to the jig, and a middling for grinding from the fifth and sixth hutches.

The size of the openings in the screens with which the tray is clothed is dependent on the size and quality of the feed to be handled and can only be determined by trial. "Ton-cap" screen has been used with very satisfactory results in the Hancock jigs. It outwears other forms of screen and is much easier to keep open.

III. RICHARDS PULSATOR CLASSIFIER SYSTEM.

a. Experimental work.—During the summer of 1907, a six-pocket Richards pulsator classifier of the inverted type was installed in Section No. 1.[1] The classifier was preceded by a pyramid-shaped desliming tank,

Richards Pulsator Classifier System Treating Middling from Hancock Jig Crushed through 2.5 mm. Round Hole.

PRODUCT	RATE PER 24 HOURS		ASSAY		PER CENT. OF TOTAL	
	Pulp	Solids	Copper	Insoluble	Solids	Copper
	Gallons	Pounds	Per Cent.	Per Cent.		
Feed...	107,900	256,000	1 35	.. ...	100 0	100 0
Concentrate..............	7,100	30,000	6.57	29 0	11 7	56 9
Tailing	54,500	71,400	0 34	.... .	27 9	6.9
Middling.	91,600	75,400	0.60	.... .	29 5	13.0
Vanner feed and slime ...	219,800	79,200	1.02	....	30 9	23.2
Total product	373,000	256,000	1.35	....	100.0	100.0

Tons treated per 24 hours by system......... 128
Tons treated per 24 hours by classifier........ 117
Gallons water used per ton treated by system.. 2,070
Gallons water used per ton treated by classifier...................... 1,410

which removed part of the slime from the pulp. The spigot discharge of this tank was fed directly to the classifier. This tank received the middling from the Hancock jig crushed through 2.5 mm. round hole, but not any of the undersize of 2.5 mm. from the mine fines or coarse-crushing section. The feed was therefore comparatively low grade, assaying 1.35 per cent. of copper. A test was made on this system during September 1907; as the flow sheet and complete details of this test are given in Richards's *Ore Dressing*, Vol. III., p. 1392, only a summary of the test will be given here.

The classifier made six spigot products, each of which was treated on

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[1] For a description of this classifier see paper Development of Hindered-Settling Apparatus, by Dr. Robert H. Richards, *Trans.*, XLI, 396 (1910).

a Wilfley table. The spigot products are numbered from 1 to 6, No. 1 being the finest and No. 6 the coarsest product. The No. 1 and No. 2 spigots each contained slime. The tables treating the No. 1 and No. 2 spigots produced clean concentrate, middling and head-water slime to vanners and tailing to a 50-mesh single-belt Callow screen, the oversize being tailing to waste and the undersize going to vanners. The No. 3 and No. 4 spigot tables produced clean concentrate, middling to vanners and tailing to waste. The No. 5 and No. 6 spigot tables produced clean concentrate and middling for grinding. The results of the test are summarized in the table on p. 218.

Shortly after the foregoing test was completed a second test was made on the pulsator-classifier system, using as feed the product of 5-ft. Huntington mills crushing through 1.25 by 10 mm. slot screens. The flow sheet of this test was the same as for the preceding test except that the pulsator-classifier spigots, 4, 5 and 6 (coarse) were combined after leaving the classifier and treated on one Wilfley table instead of being treated separately.

Richards Pulsator Classifier System Treating Middling from Evans Jigs Crushed through 1.25 by 12 mm. in Huntington Mills.

PRODUCT	RATE PER 24 HOURS		ASSAY		PER CENT. OF TOTAL	
	Pulp	Solids	Copper	Insoluble	Solids	Copper
	Gallons	Pounds	Per Cent.	Per Cent.		
Feed.	207,300	374,200	1.32	....	100.0	100.0
Concentrate...............	5,900	26,070	6.98	26.5	7.0	36.9
Tailing.	61,900	128,530	0.44	....	34.4	11.5
Middling	44,500·	146,120	0.92	.. ..	39.1	27.4
Vanner feed and slime.....	307,700·	73,480	1.64	.. ..	19.5	24.2
Total product.......	420,000	374,200	1.32	....	100.0	100.0

Tons treated per 24 hours by system........................ 187
Tons treated per 24 hours by classifier..... 171
Gallons water used per ton treated by system................ 1,140
Gallons water used per ton treated by classifier........................ . 760

Further experiments were made on the treatment of the 2.5 mm. undersize resulting from the crushing of the Hancock-jig middling by the pulsator-classifier system in which the last three spigots of the classifier (coarse) were combined and treated on one Wilfley table. This amounted to making four instead of six classifications, a procedure to which Dr. Richards was opposed from the first.

The classifier system as installed in Section No. 1 had shown so much better results than the Evans-jig and unclassified table-feed system which

was in use at that time in the mill that it was decided to remodel one section (No. 4) of the mill and install the pulsator-classifier system.

b. Richards pulsator classifier system installed in Section No. 4.—The flow sheet of this remodeled section will be outlined briefly. The crude ore after being crushed through 1.5-in. round hole by Blake crushers was passed over a series of three screens: trommels 3 by 6 ft.—$\frac{7}{8}$-in. round hole, and 8 mm. ($\frac{1}{3}$-in.) round hole and 2.5 mm. ($\frac{1}{10}$-in.) round hole, respectively. The oversize of the $\frac{7}{8}$-in. and 8 mm. trommels was combined and treated on a 42 by 30-in. Woodbury jig. This jig produced coarse concentrate, a hutch product which was sent to the Hancock jig, and a middling which was divided into coarse and fine by screening and crushed through two sets of rolls (15 by 28 in.), coarse and fine respectively, the products of the rolls being returned to the head of the system. The oversize of the 2.5 mm. trommels was treated on one Hancock jig (20-ft. tray). This jig made a coarse concentrate from the first three hutches, a return middling to the jig from the fourth hutch, and a middling to be crushed from the fifth and sixth hutches. This Hancock-jig middling was crushed in two sets of rolls, through 2.5 mm. round hole, and fed to two six-compartment Richards pulsators in which only four compartments were used. The four classified products were fed to four sets of Wilfley tables. The four tables treating the first two spigots separately—fine material—produced a concentrate, a middling to vanners which in turn produced concentrate and tailing, a tailing which was treated on a 50-mesh Callow screen, the oversize being tailing and the undersize going to vanners which produced a concentrate and a tailing, and a head-water slime which went to the slime division for treatment. The two tables treating the third spigot made a concentrate, a tailing to waste, head water to the slime division, and a middling which was ground in the Huntington mills. The two tables treating the fourth or coarse spigot made a concentrate, a middling and tailing to be ground in the Huntington mill, and a head water (carrying practically no solid matter) to waste. The undersize of the 2.5 mm. trommels containing the original mine fines and the fines from the coarse-crushing section was treated in two six-compartment pulsator classifiers, but four compartments being used. The flow sheet of this classifier system is identical with the one just described except that the fourth spigot, instead of going directly to tables, was treated on a Richards pulsator jig. This jig made a concentrate and a middling, the middling being sent directly to two Wilfley tables, which made a concentrate, a middling and tailing to be ground in the Huntington mills, and a head water to waste. The middling was ground in 5-ft. Huntington mills through 1.25 by 12 mm. slots and treated in one six-compartment pulsator classifier, only four compartments being used. The two tables treating the first two spigots (fine) separately made

a concentrate, a middling to vanners which produced a concentrate and a tailing, a tailing to 50-mesh Callow screens, the oversize being tailing and the undersize going to vanners which made concentrate and tailing a head-water slime to the slime division. The table treating the third spigot made a tailing and head water to waste and a middling which was returned to the Huntington mills. The table treating the fourth spigot made a concentrate, a middling which was returned to the Huntington mills, and a tailing and head water to waste. The total slime, consisting of the overflow of the desliming tanks preceding the pulsator classifiers and the head water slime from the Wilfley tables as noted above, was treated in the slime division. This division consisted of 30 8-ft. Callow cones for thickening the slime, the spigot of the cones going to 16 round tables, 18 ft. in diameter, and one Deister No. 3 slime table, and the overflow going to waste. These tables made concentrate and tailing. This section had a capacity of 500 tons per day.

The new section did not do as good work as had been anticipated, the tailing being little, if any, better than that produced in the old sections and the concentrate more siliceous. The failure of the section is attributed largely to the presence of the original mine fines in the pulsator-classifier system, this material being rich in fine free mineral. It should be noted that this product was not included in the pulps that were treated experimentally by the pulsator-classifier system in Section No. 1. Another condition which had a decided effect upon the results was the constant variation in the quantity and quality of the mine fines, a condition with which the pulsator classifiers could not successfully cope on account of their extreme sensitiveness to changes in the feed. It must be borne in mind that the Great Falls mill was built before the day of independent crushing plants, and although the value of separate crushing has long been recognized at this mill, one thing and another has interfered with the carrying out of the plans for an independent crushing unit. The mine-run ore is fed direct to each section which does its own crushing, and consequently the quantity and quality of the undersize from the original 2.5 mm. trommels varies immediately and considerably with the variations in the character of the ore.

Several alterations were made in the section, such as the substitution of six classifications in place of four in the pulsator classifiers, the elimination of the pulsator jig, the mixing of the original 2.5 mm. undersize with the 2.5 mm. middling product and Huntington mill product, and the substitution of Harz jigs for Wilfley tables in the treatment of the coarser spigots of the pulsator classifiers. None of these changes, however, increased the efficiency of the section to any extent.

IV. WOODBURY CLASSIFIER AND JIG SYSTEM.

Experiments made in 1905 on Woodbury classifiers and jigs had shown such favorable results that it was decided to install this system in Section 4, to be run in parallel with the Hancock jig—Richards pulsator-classifier system. A Woodbury classifier and two jig compartments connected in series, known as an 8 mm. unit, were set up in parallel with the Hancock jig. This unit received the total 8 mm. undersize as feed. The classifier removed the slime and fine sands (undersize of about 0.4 mm.), which were sent to V tanks, the spigot product being treated on Wilfley tables and the overflow of the tank sent to the slime division. In addition, this classifier made a cup and a hutch concentrate and a classified middling, which was fed to the first jig. This jig made a cup and a hutch concentrate and a middling, which passed over the second jig. The second jig produced a cup concentrate, a hutch product which was treated on a Wilfley table, and a middling which was crushed through a 2.5 mm. round hole by two sets of rolls. The total 2.5 mm. was split into halves, one part going to a Richards pulsator system similar to that described and the other part being treated by the Woodbury 2.5 mm. system. The Woodbury 2.5 mm. unit consisted of a Woodbury classifier followed by four Woodbury jig compartments (48 by 30 in.) arranged in series. The classifier removed the slime and fine sand, which were sent to a V tank, the spigots of the tank being treated on Wilfley tables and the overflow going to the slime division. The classifier also produced a hutch concentrate and a classified middling, which went to the first jig. This jig produced a hutch product, which was treated on a Wilfley table, and a middling which passed to jig No. 2. Jig No. 2 made a hutch product for table work, a middling to be ground in Huntington mills and a middling which went to jig No. 3. This jig made the same products as No. 2, passing a middling to the No. 4 or last jig. The No. 4 jig made a hutch product for table work and a tailing. The Wilfley tables treating the classifier overflows made a concentrate, a middling for vanner treatment, the vanners making a concentrate and tailing, a tailing and a head-water slime to the slime division. The tailing was screened over a 50-mesh Callow screen, the oversize going to waste and the undersize to vanners which made concentrate and tailing. The Wilfley tables treating the hutch products of the jigs made a concentrate, a middling to Huntington mills, and a tailing to waste. After considerable experimental work and change in the flow sheet this system was also abandoned as not being applicable for the most efficient treatment of the Butte ores.

V. THE GREAT FALLS SYSTEM OF CONCENTRATION.

Two classifying systems, the Richards pulsator and the Woodbury, had now been tried out and had not proved satisfactory. During the summer

of 1911 the Richards-Janney classifying system had been tried in the Washoe concentrator at Anaconda, which mill receives about the same ore as the Great Falls mill, and had not given satisfactory results. The experimental work that had been carried on had proved quite conclusively that the Hancock jig is more efficient for the concentration of the Butte ores than the Evans jig, that the Harz type of jig will not produce a satisfactory tailing on account of the fine free mineral which the jigging action throws into the tailing, that a reciprocating table is the best form of concentrator for the production of tailing, and that the feed to any concentrating machine should be so prepared, either by classifying or sizing, that it will contain only those grains of mineral and gangue material which can be treated efficiently by the machine in question. Classification was preferred to screening in preparing for treatment material finer than 4 mm. on account of its simplicity and lower cost. The problem then became one of either finding or developing a suitable form of classifier. As some of the best classifiers on the market had been tested and found wanting, it seemed that a classifier would have to be developed for the particular work to be done. This classifier had to be a flexible one which could handle efficiently a feed which varied considerably in quantity and quality.

During the summer of 1910 experiments had been carried on to determine the adaptability of the Richards pulsator classifier of the direct type. Some difficulty was experienced in keeping the slime contained in the feed out of the spigot products, and to overcome this difficulty Dr. Richards advised that we use a conical tank to deslime partly the feed to the classifier. He suggested the use of a short sorting column at the base of the cone in which a pulsating rising current of water was to be introduced. It was found later that this pulsation was of no advantage and a plain rising current was used. This deslimer gave very satisfactory results, removing all of the slime from the feed to the Richards classifier.

At this time the Hancock jigs were receiving as feed the total undersize of about ⅜ in., the slime being partly removed by a plain V settling tank before the material was fed to the jigs. The presence of the slime and fine sand in the feed was detrimental to the work of the jigs and it was decided to try one of these desliming cones ahead of the jig, not only to remove the slime from the jig feed but also the fine sand. This installation gave very satisfactory results, the work of the jig being improved considerably; a large amount of table feed was sent directly to finishing tables, where it belonged.

Having obtained such satisfactory results from these desliming cones, which were virtually classifiers, it was decided to develop this machine with the hope of producing an efficient hindered-settling classifier. The experimental work, which extended over a period of a year or more, re-

sulted in the development of an extremely simple classifier, which gave practically perfect hindered-settling products.[1]
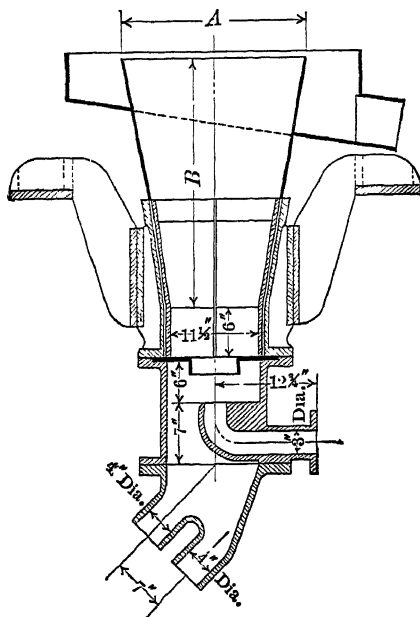
No. 1 and No. 3, A—7 ft. 0 in.; B—4 ft. 3 in.
No. 2,   A—2 ft. 4 in.; B—3 ft. 6 in.
No. 4,   A—2 ft. 1 in.; B—3 ft. 6 in.
No. 5,   A—1 ft. 11 in.; B—2 ft. 6 in.
No. 6,   A—5 ft. 0 in.; B—3 ft. 6 in.

FIG. 2.—ANACONDA CLASSIFIER.

Having developed what seemed to be an efficient hindered-settling classifier, we were now ready to design a flow sheet embodying the principles which our years of experimental work had brought out. In designing this flow sheet, the following factors were kept constantly in mind:

1. Maximum recovery of valuable minerals;
2. Simplicity of flow sheet and machines;
3. Minimum consumption of water;
4. Minimum cost of operation;
5. Maximum tonnage per unit of floor space.

a. Description of Great Falls flow sheet as installed at Anaconda.—Reference should be made to the pictorial flow sheet of the Great Falls system, Fig. 3, and to the pictorial flow sheet of the Anaconda concentrator, Sections 2–8, Fig. 4. The Anaconda flow sheet, sometimes referred to as the "Evans system," was modeled after the flow sheet used in the Great

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[1] See paper, The Anaconda Classifier, by Robert Ammon, p. 277, this volume.

Falls mill and is practically the same, the most important change being the use of Wilfley tables in place of Frue vanners for the treatment of the Wilfley table middling. The Anaconda flow sheet is the one which was replaced in Section No. 1 of the Anaconda mill by the Great Falls system.

1. *Coarse crushing and jigging division.*—In installing the Great Falls system of concentration in Section 1 of the Anaconda mill, the coarse crushing and jigging section remained practically as it was. Although the installation of an independent coarse-crushing plant has been under consideration for a number of years the opportunity for its installation has not presented itself as yet. The mine-run ore is therefore fed directly to the concentrating sections. There is one 1,000-ton ore-supply bin for each section of the mill. The ore is fed from the bin through two air-operated gates to two shaking screens with 2-in. round holes. Two men are employed as feeders, one on each shaking screen, and the usual practice is for one man to feed while the other cleans his screen. In this way a fairly uniform feed is maintained. The oversize of the shaking screens goes to one 12 by 24-in. Blake crusher set to crush through 2.5 in. The product of the crusher goes to two trommels, 3 by 6 ft., clothed with 2-in. round-hole, cast manganese-steel screens. The oversize of these screens goes to two 5 by 15-in. Blake crushers set to crush through 2-in. The undersize of the shaking screens, the undersize of the 2-in. trommels, and the product of the two smaller Blake crushers are elevated to the head of the mill, where they pass through four 1-in. round-hole trommels, 3 by 6 ft., clothed with punched sheet steel. The oversize of these screens is treated on two double-compartment coarse Harz jigs, which make a cup concentrate, a hutch product, and a middling. The middling is partly dewatered by inclined screens and passes to one set of coarse rolls, 24 by 54 in., crushing through 1 in., and from there returns to the head of the system. The water from the middling, together with the hutch product, joins the feed to the coarse Hancock jigs. The concentrate is dewatered by a shaking screen and passes up an inclined conveyor to the coarse-concentrate bins. The water from the concentrate is used to sluice the undersize of the 2-in. shaking screens. The undersize of the 1-in. trommels goes to four 10 mm. ($\frac{2}{5}$-in.) round-hole trommels, 3 by 6 ft., clothed with punched-steel screens. The oversize of these screens passes to four double-compartment Harz jigs, which make a cup concentrate, a hutch product, and a middling. The concentrate joins the concentrate from the coarse Harz jigs. The middling is partly dewatered and passes to one set of fine rolls, 24 by 54 in., crushing through 10 mm. ($\frac{2}{5}$ in.), and from there returns to the head of the system. The hutch product, together with the water from the middling, unites with the feed to the coarse Hancock jigs.

The Anaconda flow sheet in the remaining sections of the mill differs from the above in the use of but two 1-in. trommels and two 10 mm. trommels.

2. Hancock jig division.—The total undersize of the 10 mm. trommels
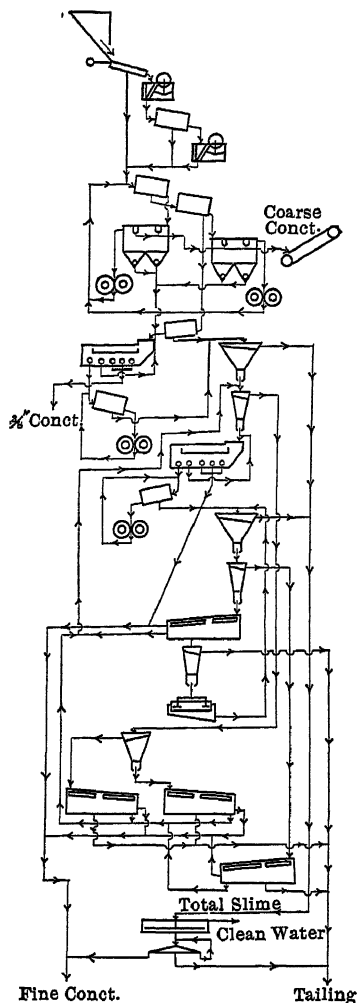
1 Ore Bin.

2 Shaking Screens, 2-in Round Hole.

1 Blake Crusher, 12 by 24 in.

2 Trommels, 2-in. Round Hole.

2 Blake Crushers, 5 by 15 in.

4 Trommels, 1-in. Round Hole.

4 Trommels, 10 mm. Round Hole.

2 2-Comp. Harz Jigs (Coarse). 2 2-Comp. Harz Jigs (Fine).

1 Set of Rolls, 24 by 54 in. (Coarse)   1 Set of Rolls, 24 by 54 in. (Fine).

8 Trommels, 4 mm. Round Hole.

2 Hancock Jigs (Coarse).   6 No. 1 Anaconda Classifiers.

6 Trommels, 4 mm. Round Hole.   6 No. 2 Anaconda Classifiers.

1 Set of Rolls, 16 by 42 in.   3 Hancock Jigs (Fine)

8 Trommels, 1.25 by 12-mm. Slots.

2 Sets of Rolls, 16 by 42 in.   8 No. 3 Anaconda Classifiers.

6 No. 4 Anaconda Classifiers.

9 Wilfley Tables (Coarse Middling).

2 No. 5 Anaconda Classifiers.

4 Huntington Mills, 6 ft.

3 No. 6 Anaconda Classifiers.

19 Wilfley Tables (Fine Primary). 12 Wilfley Tables (Coarse Primary).

23 Wilfley Tables (Secondary).

14 Dorr Thickeners, 28 by 3 ft.
35 Round Table Decks (Concrete).

Fig. 3.—Great Falls System of Concentration.   Remodeled Section No. 1, Anaconda Concentrator.

goes to eight 4 mm. ($\frac{1}{16}$-in.) round-hole trommels, 3 by 6 ft., clothed with punched-steel screens. The oversize of these screens is fed to two coarse 19-ft. Hancock jigs (trays 15 ft. long), which make a concentrate from the first three hutches, a middling from the fourth hutch, which is returned to the jig, and a middling from the fifth and sixth hutches, which is ground

through 4 mm. by one set of intermediate rolls, 16 by 42 in., the product of the rolls going to six 4 mm. round-hole trommels, 3 by 6 ft., the over-size of the trommels returning to the rolls. The concentrate passes to a set of dewatering tanks in the tank house and is known as $\frac{3}{8}$-in. concen-
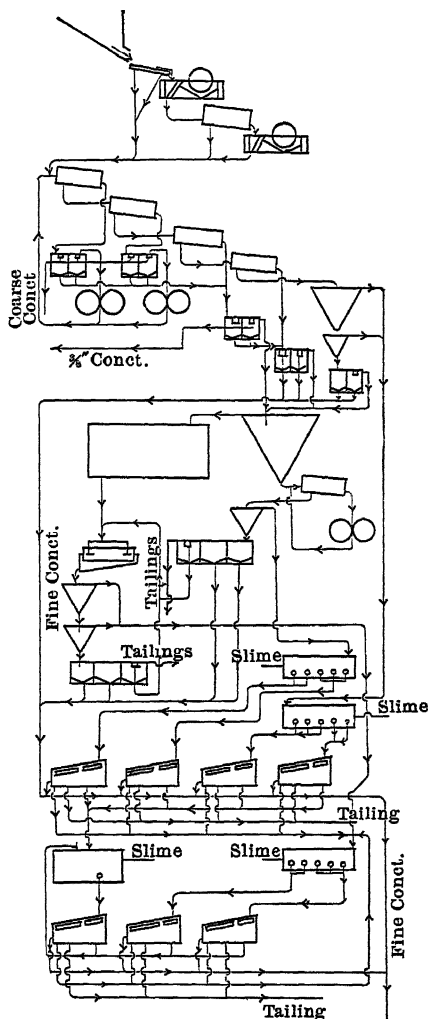
1 Ore Bin.
2 Shaking Screens, 2-in. Round Hole.
1 12 by 24 in Blake Crusher.
2 Trommels, 2-in. Round Hole.
2 5 by 15 in. Blake Crushers.

2 Trommels, 1-in. Round Hole.
2 Trommels, 10 mm. Round Hole.
4 Trommels, 5 mm Round Hole.
4 Trommels, 2.5 mm. Round Hole.
2 Coarse Jigs. 4 Fine Jigs.
1 Coarse Roll, 16 by 42 in. 1 Fine Roll, 16 by 42 in.
12 $\frac{3}{8}$-in. Jigs, 2 comp.
4 Evans 3-Comp. Classifiers.
12 Medium Jigs, 2 comp.
12 Fine Jigs, 3 comp.

2 Tanks. 2 V Tanks.

4 Trommels, 1.5 by 12 mm.

4 Evans 3-Comp. Classifiers. 2 Middling Rolls, 16 by 42 in.
3 6-ft. Huntington Mills. 12 Roll Jigs, 3 Comp.

4 Evans 3-Comp. Classifiers.

12 Huntington Mill Jigs, 3 comp. 8 V Tanks.

8 V Tanks.

{ 5 Wilfley Tables.
  8 Wilfley Tables.
  7 Wilfley Tables.
  4 Wilfley Tables.

1 Settling Tank. 8 V Tanks.

{ 1 Wilfley Table.
  3 Wilfley Tables.
  5 Wilfley Tables.

FIG. 4.—EVANS SYSTEM OF CONCENTRATION. SECTIONS NOS. 2 TO 8, ANACONDA CONCENTRATOR.

trate. These coarse Hancock jigs receive also the water from the mid-dlings and the hutch products from the Harz jigs in the coarse crushing and jigging division, as noted above. This water replaces part of the water that would otherwise have to be added in the hutch of the jigs and as a result only about 100,000 gal. of water are added in the hutch of each jig.

The overflow of water from the jigs, together with the overflow water from the jig middling dewatering tanks, furnishes about three-fourths of the water required on the fine Hancock jigs. The total undersize from the 14 4-mm. trommels passes to six Anaconda classifiers No. 1. (See Fig. 2, p. 224.) These No. 1 classifiers overflow slime (0.08 mm. to 0 quartz) which is sent directly to the slime division. The spigot discharge of these classifiers goes to the six No. 2 classifiers, which overflow 0.75 to 0.08 mm. material and discharge the 4.0 to 0.75 mm. material through the spigot. All sizes given for the Anaconda classifier products refer to the diameter of the quartz grain, the mineral grain being about one-fourth the diameter of the quartz grain. For example, a classifier product of 0.75 to 0.08 mm. material would contain quartz grains ranging in size from 0.75 to 0.08 mm. and free mineral grains ranging in size from 0.20 to 0.02 mm. This spigot discharge is treated on three fine Hancock 19-ft. jigs with 15-ft. trays. These jigs make fine concentrate from the first three hutches, middling from the fourth hutch, which is returned to the jig, and middlings from the fifth and sixth hutches, which unite and go to eight trommels, 3 by 8 ft., clothed with punched-steel screen having openings 1.25 by 12 mm. This middling product is sent to the screens before going to the rolls to remove any fine free mineral which may be present. The water overflowing from the fine jigs, together with the overflow from the middling dewatering tanks, is used in the fine middling division as wash water on the tables. The final products from this division are then: ⅜-in. concentrate from the coarse Hancock jigs, fine concentrate from fine Hancock jigs, table feed from No. 2 classifiers going directly to table division, slime from No. 1 classifiers going directly to slime division, and fine middling from fine Hancock jigs.

The equipment in the Anaconda flow sheet that was replaced by this division consists of four 5 mm. round-hole trommels receiving undersize from 10 mm. trommels in coarse-crushing division, four 2.5 mm. round-hole trommels treating the undersize of the preceding trommels, four 3-spigot Evans classifiers treating the undersize of the 2.5 mm. trommels, 12 double-compartment Evans jigs treating oversize of 5 mm. trommels, 12 three-compartment Evans jigs treating oversize of 2.5 mm. trommels, and 12 three-compartment Evans jigs treating the Evans classifier products. The overflow of the Evans classifier, consisting of slime and sand, goes to the table section. The jigs treating the oversize of 5 mm. receive also the hutch product of the Harz jigs in the coarse-jigging division. These 5 mm. oversize jigs make a cup concentrate which goes to separate tanks in the concentrate tank house and is called ⅜-in. concentrate. The hutch product from these jigs goes to the next set of jigs. The 2.5 mm. oversize jigs make a cup and a hutch fine concentrate and the 2.5 mm. undersize jigs make a hutch fine concentrate.

All three sets of jigs make a middling which passes to the roll middling division. This middling after being dewatered passes over four 3 by 8-ft. trommels clothed with slotted steel punched screens, 1.5 by 12 mm. The oversize goes to two sets of 16 by 42-in. finishing rolls, the product of the rolls being returned to the trommels. The undersize of the trommels goes to four Evans three-compartment classifiers which feed 12 Evans three-compartment jigs. The overflow from the classifier, containing slime and sand, goes to the table section. These jigs make a hutch fine concentrate, a middling to the Huntington mill division, and a tailing to waste.

3. Fine middling division.—The fine middling from the fine Hancock jigs goes to eight trommels as noted previously, the oversize of these trommels (1.25 by 12 mm.) going to two sets of 16 by 42-in. rolls, and the product from the rolls being returned to the trommels. The undersize of these trommels goes directly to eight No. 3 Anaconda classifiers, which remove the slime and send it directly to the slime division. The deslimed spigot discharge of the No. 3 classifiers passes to six No. 4 classifiers of the same type, where it is classified into finishing table feed, 0.75 to 0.08 mm., and coarse middling table feed, 1.25 to 0.75 mm. The coarse middling tables, nine in number, make a fine concentrate, a rough concentrate, and a middling tailing. The rough concentrate, containing the coarser mineral grains, is returned to the No. 2 classifiers preceding the fine Hancock jigs, where the coarser free mineral is discharged through the spigot and recovered at once by the jigs. 'The middling-tailing, being a mixture of the coarser middling and tailing grains, goes to two No. 5 Anaconda classifiers, which overflow about 30 per cent. of the material as tailing and discharge the remainder through the spigot, this product going to four 6-ft. Huntington mills of the Anaconda type, clothed with punched-steel screens 0.75 by 12 mm. The excess water contained in the tailing classifier overflow is recovered and used in the No. 4 classifiers. The Huntington mill product is delivered to the No. 3 classifiers, where the slime is at once removed. The coarse middling tables are standard 16-ft. Wilfleys of the No. 6 type, slightly modified mechanically. The riffles are spaced 1.5 in. between centers and are 0.75 in. high at the head end, tapering to $\frac{1}{8}$ in. at the concentrate end, and all but the upper eight are carried to the end of the table. The final products from this division are: Fine concentrate from the coarse middling tables, rough concentrate returned to the No. 2 classifiers, slime from the No. 3 classifiers sent directly to the slime division, and table feed overflowed from the No. 4 classifiers going to the table division.

The portion of the Anaconda flow sheet displaced by this division in Section 1 consists of three 6-ft. Huntington mills grinding the middling from the Evans roll middling jigs through 1 by 12 mm., 4 three-spigot

Evans classifiers treating the Huntington mill product, and 12 three-compartment Evans jigs treating the classifier spigots. The overflow from the Evans classifiers goes to the table section. The jigs make a fine concentrate from the hutches, a tailing to waste, and a middling which is returned to the Huntington mills.

4. Table division.—Standard 16-ft. Wilfley tables are used in this division both in the remodeled section and in the original Anaconda sections. This division receives the deslimed overflow from classifiers No. 2 and No. 4. The overflow from classifiers No. 2 containing the original mine fine material is further divided into two classes by three No. 6 classifiers, which overflow 0.35 to 0.08 mm. material and discharge through the spigot 0.75 to 0.35 mm. material. Each class goes to a set of Wilfley tables which make a concentrate, a tailing, and a middling, the latter product being returned to the No. 2 classifiers. The overflow from the No. 4 classifiers, ranging in size from 0.75 to 0.08 mm., is treated directly on a set of tables which make a concentrate, a tailing and a middling which is returned to the No. 2 classifiers. The feed to the tables is dewatered, the water being used as wash water in the table division and the slime division.

The Anaconda table division is divided into two parts. The upper table floor receives the overflow from the 2.5 mm. Evans classifiers and from the roll middling Evans classifiers. This overflow is partly deslimed in V tanks before being fed to the tables. These tables make a fine concentrate, a tailing, a middling which goes to the tables on the lower floor, and a head-water slime which goes to a square tank on the lower table floor, the spigot of the tank being treated on one table and the overflow uniting with the overflow from the upper table feed tanks and leaving the mill as slime. The lower table floor receives as feed, in addition to the upper table floor middlings, the overflow from the Huntington mill Evans classifiers. This total feed is partly deslimed in V tanks and is then treated on Wilfley tables making a concentrate, a tailing, a middling, which is returned to the table-feed tanks, and a head-water slime which goes to the square tank mentioned above. The overflow from these lower table-floor V tanks unites with the slime leaving the mill.

5. Slime division.—The slime division is an experimental plant and has a capacity of about 50 tons of slime per day, which is about 20 per cent. of the total slime produced in the remodeled section. This plant has now been in operation for about one year; after considerable experimental work, in which most of the leading slime concentrators on the market were tested, it was finally decided to adopt the concrete-and-steel round table. These tables are 18 ft. in diameter and have a slope of deck surface of 1.25 in. per foot.[1] The slime pulp, having a density of 2 per cent.

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[1] See paper, Concentration of Slime at Anaconda, by Ralph Hayden, p. 239, this volume.

of solids, was originally thickened in 8-ft. Callow cones, operated to settle 95 per cent. of the solids in a product having a density of about 9 per cent. These cones did very good work, but have now been discarded in favor of 28-ft. diameter Dorr thickening tanks 3 ft. deep. These latter tanks, being larger units and flat bottomed, have the advantage over the Callow cones of a considerably lower cost of installation. These 3-ft. deep thickeners will probably be installed in stands of four tanks, one above the other.

A slime plant, consisting of Dorr thickeners and multiple-deck concrete-and-steel round tables, to treat the entire slime pulp from the eight sections of the mill, will undoubtedly be built this year. At the present time this slime pulp is settled in large ponds, the recovered solids being briquetted with fine first-class ore screenings and fine concentrate and treated in the blast furnaces.

b. Special features of the Great Falls flow sheet.—A great deal of experimental work was carried out during the development of this flow sheet and some of the more interesting experiments will be described.

1. Development of Hancock jig flow sheet.—One of the principles which was adhered to in designing the flow sheet was to treat on a given machine only that class and size of material which could be most efficiently treated by that machine. Thus it became necessary early in the work to determine the minimum size of free-mineral grain which can be recovered by the Hancock jig. This was investigated by screen sizing samples of the Hancock jig concentrate and middling taken during regular operation, and then sorting each screen-sized product into a true middling and true concentrate of given grades. This work showed conclusively that all mineral grains finer than 90 mesh (0.17 mm.) should be eliminated from the feed to the jig. Above this size the recovery of the copper which was present in the feed in the form of free mineral ranged between 95 and 100 per cent., but just at and below this limiting size the recovery dropped off abruptly, being only about 50 per cent. for mineral grains ranging in size between 90 and 120 mesh (0.17 to 0.10 mm.). As the Anaconda classifiers produce practically perfect hindered-settling products, the smallest quartz grain contained in the jig feed would be 0.17 by 4 = 0.68 mm., if we limit the smallest mineral grain to 0.17 mm. All of the quartz grains finer than 0.68 mm. would then be thrown off by the classifier and pass into the overflow together with the mineral grains finer than 0.18 mm. It so happened that it had previously been determined that the largest tailing grain should be about 0.75 mm., which size practically coincided with the maximum size grain in the overflow of the classifiers preceding the Hancock jigs. Thus it followed that this classifier overflow should be put on machines producing a tailing. It was found advisable to divide the Hancock jig feed, ranging in size as it did from 0.10 to 0.75 mm., into two parts by screening it over 4 mm. round-hole

trommels. The jigs did better work on the more closely sized feed and it also enabled us to introduce a more graded crushing by crushing the middling from the coarse jigs through 4 mm. in one set of rolls and the middling from the fine jig through 1.25 mm. in two other sets of rolls. An investigation was also carried on to determine whether the oversize of 4 mm. resulting from the first passage through the rolls should be returned to the coarse jigs or further crushed at once through 4 mm. Sorting tests made on this product showed the presence of such a small amount of free mineral that the jigging would not pay, therefore it was decided to use a closed circuit of rolls and 4 mm. trommels, and to crush all of the middling through 4 mm. before subjecting the product to concentration.

2. Development of finishing table flow sheet.—It having been determined that a reciprocating table is the best form of concentrator on which to produce a tailing and that the maximum tailing grain was to be about 0.75 mm., the next point to be investigated was the minimum size mineral grain which can be recovered by the reciprocating table, this table being in this particular case a standard 16-ft. Wilfley. This investigation was carried on for the most part in the testing laboratory on a small size Wilfley table. The limiting grain was found to be approximately 0.025 mm. in diameter, which would carry with it in the hindered-settling classifier product a grain of quartz approximately 0.08 mm. in diameter (0.08 mm. is equivalent to a 200-mesh screen opening). This meant that all slime must be eliminated from the table feed,—the term "slime" being rather loosely used to apply to all material, both granular and colloidal, finer than 0.08 mm. quartz and 0.025 mm. mineral. As a matter of fact this slime contains about 65 per cent. of granular material and 35 per cent. of colloidal material.

It will be noted that while the overflow of classifiers No. 2 is further classified into two products, 0.75 to 0.35 mm. and 0.35 to 0.08 mm., before being subjected to table treatment, the overflow of the No. 4 classifiers, consisting entirely of crushed low-grade middlings and ranging in size from 0.75 to 0.08 mm., is treated directly on the tables without further classification. This procedure is also the result of careful investigation, the experimental work having proved conclusively that while the further classification of the lower-grade material was of no advantage, the classification of the richer material increased the recovery on the tables by about 7 per cent.

3. Treatment of coarse middling table products.—As has been noted, it was originally intended to make no tailing coarser than 0.75 mm. However, when tests were made on the 500-ton unit, which was installed at Great Falls, it was found that the middling from the coarse middling table contained a considerable amount of tailing material, although

ranging in size from 0.75 to 1.25 mm. During the first test this entire product had been returned to the Huntington mills for crushing, but this so congested the middling section that the tonnage to the unit had to be materially reduced. During the second test a tailing was cut out from the product as it left the tables. This resulted in the elimination at this point in the system of 22.2 per cent. of the solids in the feed to the entire 500-ton unit assaying 0.43 per cent. of copper. It was then suggested that possibly more tailing material could be produced by the classification of this middling product, the overflow of the classifier being tailing. A laboratory test was made in which some of this middling product ranging in size from 2.0 to 0.8 mm. was treated in a hindered-settling classifier. Six products were drawn off through the spigot, the one settling against the strongest rising current being called No. 1 and the one settling out against the weakest rising current being designated as No. 6.

Classification of Middling from Coarse Middling Table.

SPIGOT No.	TOTAL SOLIDS		ASSAY COPPER			TOTAL COPPER	
	Individual	Cumulative	Individual	Cumulative		Individual	Cumulative
				Down	Up		
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1	8 8	8.8	0 98	0 98	0 40	21 0	21 0
2	15.9	24.7	0 57	0.70	0 35	22 4	43 4
3	18.8	43.5	0 36	0.56	0.30	17.1	60 5
4	37.4	80.9	0 30	0 44	0 28	27.6	88 1
5	12.2	93 1	0 25	0 41	0.25	7 9	96 0
6	6 9	100.0	0 22	0 40	0 22	4 0	100 0
Total	100.0	....	0 40	. .	. .....	100 0	.... .

The first spigot contained the coarser middling grains and a few coarse quartz grains; the second spigot contained finer middling and tailing grains, while the last spigot contained for the most part only tailing grains. There were no free-mineral grains in the original middling fed to the classifier. The preceding tabulation shows that it is possible to discard in the tailing classifier 56.5 per cent. of the solids (Spigots 4+5+6), assaying 0.28 per cent. of Cu, as tailing, and to send to the Huntington mills only 43.5 per cent. of the solids, assaying ·0.56 per cent. of Cu. In practice we have never been able to make a tailing as low as 0.28 per cent. of Cu at this point. In the third test at Great Falls on a 500-ton section this classifier tailing contained 25 per cent. of the total solids fed to the section and assayed 0.42 per cent. of Cu.

The rough concentrate from the coarse middling tables contains the coarser grains of free mineral and the finer middling grains. Experiments

were made in which this product was treated on a second set of Wilfley tables. The results, however, of this treatment were not at all satisfactory, the secondary tables making a low-grade concentrate and a tailing altogether too rich in copper. The next scheme tested was to treat this product in a classifier, with the idea of drawing off the coarser mineral grains through the spigot as a concentrate and overflowing the finer material and tailing grains for table treatment. This worked out very well on a small scale, but it was found to be impractical to produce a concentrate from a classifier in large-scale operations. The next set of experiments was made in the laboratory and consisted of a classifying and jigging flow sheet, the spigot of the classifier treating the rough concentrate containing the coarser mineral and middling grains being jigged and the overflow sent to tables. This flow sheet gave very good results and was finally adopted, the rough concentrate being returned to the No. 2 classifiers preceding the fine Hancock jigs, where the coarser mineral and middling grains would pass through the spigot and on to the jig, the mineral grains being saved at once as concentrate and the middling grains going to the roll system for further grinding. The fine mineral grains and tailing grains being overflowed from the No. 2 classifiers, pass at once to the finishing table system.

4. *Results of test made on 500-ton section at Great Falls.*—A test made during October, 1911, on a 500-ton unit at the Great Falls mill gave the following results:

Test Made at Great Falls Plant on 500-Ton Section.

PRODUCT	Solids per 24 Hours	ASSAY		Copper per 24 Hours	PER CENT. OF ORIGINAL	
		Copper	SiO₂+Al₂O₃		Solids	Copper
	Pounds	Per Cent.	Per Cent.	Pounds		
Ore to section.......	986,075	3.51	73.3	34,651	100.0	100.0
Coarse concentrate ..	67,000	13.83	24.5	9,266	6.8	26.7
Fine concentrate.....	166,595	10.00	13.7	16,652	16.9	48.1
Slime concentrate....	42,610	9.49	53.8	4,044	4.3	11.7
Total concentrate..	276,205	10.85	22.5	29,962	28.0	86.5
Mill tailing..........	469,120	0.40	96.3	1,900	47.6	5.5
Slime tailing........	211,745	1.01	87.0	2,149	21.5	6.2
Total tailing......	680,865	0.59	93.4	4,049	69.1	11.7
Overflow loss........	29,005	2.21	82.5	640	2.9	1.8
Total products....	986,075	3.51	73.3	34,651	100.0	100.0

Following are some screen analyses of the tailings made during the tests of the system at Great Falls:

Screen Sizing Analyses of Tailings.

Screen Size Millimeters	Secondary Wilfley Tables			Coarse and Fine Primary Tables			No. 5 Classifier Overflow			Slime Plant			Total Mill Tailing			Total Mill and Slime Tailing		
	Solids Per Cent.	Assay Cu. Per Cent.	Cop- per Per Cent.	Solids Per Cent.	Assay Cu. Per Cent.	Cop- per Cent.	Solids Per Cent.	Assay Cu. Per Cent.	Cop- per Per Cent.	Solids Per Cent.	Assay Cu. Per Cent.	Cop- per Per Cent.	Solids Per Cent.	Assay Cu Per Cent.	Cop- per Per Cent.	Solids Per Cent.	Assay Cu. Per Cent.	Cop- per Per Cent.
On 2 38	.	.	......	.	.	......	0 10	0.66	0 13	.....	.....		0 06	0.66	0 08	0.04	0.66	0 03
On 1 41	.	.	......	.	.	......	6 84	0 52	8.35	.....	.....	.	3 84	0 52	5.44	2 65	0 52	2.16
On 0.84	2 31	0.23	2 39	1 65	0 43	1 74	45.42	0.43	46.49	.....	.....	.	26 41	0 42	30 99	18 11	0 12	12.31
On 0.50	15.58	0.20	13.66	8.00	0.34	6.88	40 03	0 30	36 82	.....	.	.	28 08	0 36	27.54	19.25	0 36	10 93
On 0 35	35 39	0 21	33.42	10.23	0 44	11 29	7.61	0.45	8 21	.....	.	.	15 70	0 30	13 09	10 77	0 30	5 20
On 0 17	28 64	0 19	24 32	57.42	0.30	44 38	...	......	......	.....	.	.	17 23	0 25	12.03	11.82	0 25	4.78
On 0.07	14 98	0 14	9.08	19 54	0.37	18 18	......	.....	......	16 62	0 32	1 40	7 31	0 24	4 78	10.23	0 28	4 55
Through 0 07	3 10	1 25	17.18	3.16	2.10	17.53	......	.....	.	83 38	1 38	95 60	1.37	1 60	6 05	27.15	1 38	4 04
Total	100.00	0.23	100 00	100 00	0 39	100 00	100 00	0 42	100 00	100 00	1 20	100 00	100 00	0 36	100 00	100 00	0 62	100 00

VI. COMPARATIVE DATA OF OPERATION OF GREAT FALLS AND EVANS
SYSTEMS AT ANACONDA.

The Great Falls flow sheet was installed in Section No. 1 of the Ana-
conda mill during the summer of 1912. The remodeled section was first
put into operation during October, 1912, and by November 1 was running
normally, treating its rated capacity of 1,500 tons per day.

The data contained in the accompanying table for the Great Falls
system, remodeled Section No. 1, were obtained from a test made during
March, 1913, and those for the Evans system, Sections 2 through 8, were
taken from the metallurgical sheets covering the first six months of 1912.
The tailing obtained during the test made on Section 1 assayed 0.48
per cent. of copper and the tailing for the first six months of operation
of this section also averaged 0.48 per cent. of copper, so that we may say
that the test conditions, which as a matter of fact were the regular op-
erating conditions for the section, did not favor the section at all. Also
by referring to tabulation on p. 234 entitled " Test Made at Great Falls
Plant on 500-Ton Section," we find a mill tailing assaying but 0.40 per
cent. of copper. It is thought that the mill tailing for the remodeled
section at Anaconda will be lowered in the near future to from 0.40 to
0.45 per cent. of copper as the operators become more familiar with the
flow sheet. It should be noted that the ore treated during the test on
the remodeled section happened to be of a lower grade than the usual
run of concentrating ore at Anaconda,—2.98 per cent. of copper as against
an average of 3.27 per cent. for Sections 2 through 8. Other conditions
being constant, this lower copper content of the ore would of course
lower the recovery.

The slime produced in the mill is not being concentrated at present,
but a plant consisting of Dorr tanks and concrete multiple-deck round
tables is being planned and will soon be erected for the treatment of all
of the mill slime. With the complete treatment the remodeled section
shows a net increase in recovery of copper of 3.0 per cent. over the original
mill. The recoveries of silver and gold follow very closely that of the
copper. The recoveries of copper in the slime from the two systems are
based upon the results of treating these slimes in the 50-ton experimental
slime division.

The Great Falls system as originally installed at Anaconda includes
a "fine sand" division equipped with 20 8-ft. Callow tanks followed by
16 Wilfley tables. This division receives the total classifier overflow from
the remodeled section. This pulp is thickened in Callow tanks and the
thickened pulp fed to the Wilfley tables. These tables make concentrate,
middling, tailing, and head-water slime. The middling is returned
to the tables and the head-water slime unites with the overflow from the

Comparative Data: Great Falls System and Evans System at Anaconda.

PRODUCTS	GREAT FALLS SYSTEM, REMODELED SECTION NO. 1					EVANS SYSTEM, SECTIONS 2 THROUGH 8				
	PER CENT. OF TOTAL		ASSAY			PER CENT. OF TOTAL		ASSAY		
	Solids	Copper	Cu	Insol.ᵃ	FeO	Solids	Copper	Cu	Insol.ᵃ	FeO
			Per Ct.	Per Ct.	Per Ct.			Per Ct.	Per Ct.	Per Ct.
Original ore	100.0	100 0	2.98	68 8	14 2	100.0	100.0	3 27	66.4	13 9
Total concentrate	32.9	77 0	7 00	25.8	35.3	32 3	77.0	7.78	25 3	34 2
Slime	19 6	14 1	2.14	…	…	17 2	11 2	2 13	…	…
Overflow loss	2 4	1.6	2 03	…	…	3.4	2 2	2 13	…	…
Tailing loss	45.1	7.3	0 48	…	…	47 1	9 6	0 67	…	…
Total loss	47.5	8 9	0 56	…	…	50 5	11 8	0.76	…	…
Total products	100.0	100 0	2 98	…	…	100 0	100 0	3.27	…	…

Final Summary: Results When Slime is Concentrated.

	GREAT FALLS SYSTEM, REMODELED SECTION NO. 1		EVANS SYSTEM, SECTIONS 2 THROUGH 8	
	Per Cent. of Total		Per Cent. of Total	
	Solids	Copper	Solids	Copper
Original ore	100 0	100.0	100 0	100.0
Mill concentrate	32 9	77 0	32.3	77.0
Slime concentrate	3.1	8.5	2.8	5.6
Total concentrate	36 0	85.5	35.1	82 6
Mill loss	47.5	8 9	50.5	11.8
Slime tailing	16.5	5 6	14.4	5 6
Total loss	64.0	14 5	64.9	17.4
Total products	100.0	100.0	100.0	100.0

ᵃ "Insoluble" is silica plus alumina.

Callow tanks and leaves the plant as slime. This sand division has been recovering about 3.0 per cent. of the copper in the original ore, so that if we include the work of this fine-sand division the net mill recovery in the table on p. 237 would be 80 per cent. instead of 77 per cent. However, we have found that a reciprocating table is not well adapted to the treatment of this fine sand (approximately 200 mesh and finer), and as soon as the slime plant is completed this fine-sand division will be discarded. I have therefore eliminated this division from this paper, and all of the classifier overflow from the remodeled section is included in the product termed "slime."

The increase in the recovery of copper in the remodeled section over that of the other sections using the Evans flow sheet is due for the most part to the saving of the fine free mineral which is lost in the tailing from the Evans jigs and the fine free mineral which is lost in the Wilfley table tailing of the Evans system. Another decided advantage of the Great Falls flow sheet is its capability of producing a very high grade of fine concentrate without any sacrifice in the recovery of the valuable minerals. This advantage is due entirely to the excellent classification resulting from the hindered-settling classifiers preceding the Wilfley tables. The system is operated at Anaconda to produce a fine concentrate containing about 25 per cent. of insoluble, as this grade of concentrate meets the present smelter requirements. However, during tests made at the Great Falls plant fine concentrate running as low as 10 to 15 per cent. of insoluble was produced in a 500-ton section. The insoluble in the coarse concentrate cannot be reduced below 18 to 22 per cent. on account of the fact that this amount of insoluble is combined with the mineral.

Concentration of Slimes at Anaconda, Mont.

BY RALPH HAYDEN, ANACONDA, MONT.

I. INTRODUCTION.

In the wet concentration of ores there is always a certain amount of very fine material with which to contend, and which invariably carries such quantities of value that the treatment of it is worth while, commercially. This fine material usually contains that portion of the ore which hinders the separation of the mineral values: namely, the colloidal material resulting from the decomposition of certain constituents of the ore.

The object of this paper is to show what efforts have been made to solve the slime problem as it occurs in the concentrator of the Washoe

Reduction Works at Anaconda, Mont., where the copper sulphide ores from Butte are treated.

Before entering upon the discussion of the problem, it is necessary to define the material under consideration, in order that the problem may be apparent, and that due consideration may be given to the effort made in trying to solve it.

This fine material is termed slime; but as slime is a widely used and much misused term, further definition is necessary. Local conditions in individual mills determine the constitution of slimes. The material which is called slime in one mill may not be termed such in another mill. Different treatments of the same ore may produce slimes with different characteristics. This latter point is shown by the fact that the concentrator at Great Falls, treating the same general class of ores as is treated at Anaconda, produces a slime containing more copper and a larger proportion of granular material. As a result of this difference in granular content, any settling tank will handle about twice as much Great Falls slime as Anaconda slime, when operated to the same efficiency. Therefore, when conclusions from experiments conducted at Great Falls were applied to Anaconda slime they did not hold good in all cases.

II. DEFINITION OF ANACONDA SLIME.

Anaconda slime may be defined as that portion of the Butte sulphide copper ores, concentrated by the Anaconda practice, which results from the breaking down of the ore in the mine and the crushing and regrinding of the ore in the mill; and which overflows the settling tanks receiving the overflow of the hydraulic classifiers, plus that which is contained in the head waters of the tables treating the spigot product of these tanks.

The constitution of the slime under consideration is determined largely by the settling capacity of the tanks and by the amount of table head water cut from the table tailings. But as the operating conditions are uniform, there is little variation in the slime produced.

III. THE SOURCE OF SLIME.

The two flow sheets, Figs. 1 and 2, show the source of slime in Section No. 1 and Sections Nos. 2 to 8, respectively.

Section No. 1 was remodeled during the summer of 1912, according to data obtained from experiments conducted at Great Falls. The classifier installed consists of a single pocket and has center feed and peripheral overflow. It was developed at Great Falls and is known

as the Anaconda single-pocket classifier.[1]   Two sets of these are used to remove the slime from the ore before further classification.   Six No. 1 classifiers remove the original slime and also what is made in the crushers and in the coarse, fine, and intermediate rolls.   Seven No. 3 classifiers remove the slime produced by the middling rolls and the Huntington mills.   The overflows of these two sets of classi-

Undersize of 4 mm Trommels

6 B. & M. Classifiers No. 1

To Classifiers No. 2
Undersize of 1.25 x 12 mm Trommels

8 B. & M. Classifiers No. 3

To Classifiers No. 4

20-8'0" Callow Cones

16 Wilfley Tables

Part to Slime Plant

24 Callow Cones

Excess to Ponds

8-18'0" Deck Round Tables

——— Final Slime to Slime Plant or to Ponds
— — — Slime Producing Products
——— Sand Products
———C= Concentrates to Smelter
———T= Tailings to Waste
— · — · · Clean Water for Re-use

FIG. 1.—SOURCE OF SLIME IN SECTION No. 1.

fiers are combined and go to 20 Callow tanks, where the coarser sand is removed through the spigot and treated on 16 Wilfley tables. This spigot product carries down some of the colloidal material, which is removed as head water of the tables, and which joins the overflow of the 20 Callow tanks, the combination constituting the slime from Section No. 1.

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[1] See Fig. 2 of paper by A. E. Wiggin, The Great Falls System of Concentration, p. 224, this volume.

The two sets of classifiers, No. 1 and No. 3, use sufficient hydraulic water to·just give a clear water spigot, so that no colloidal material is treated on any machine in the section proper.
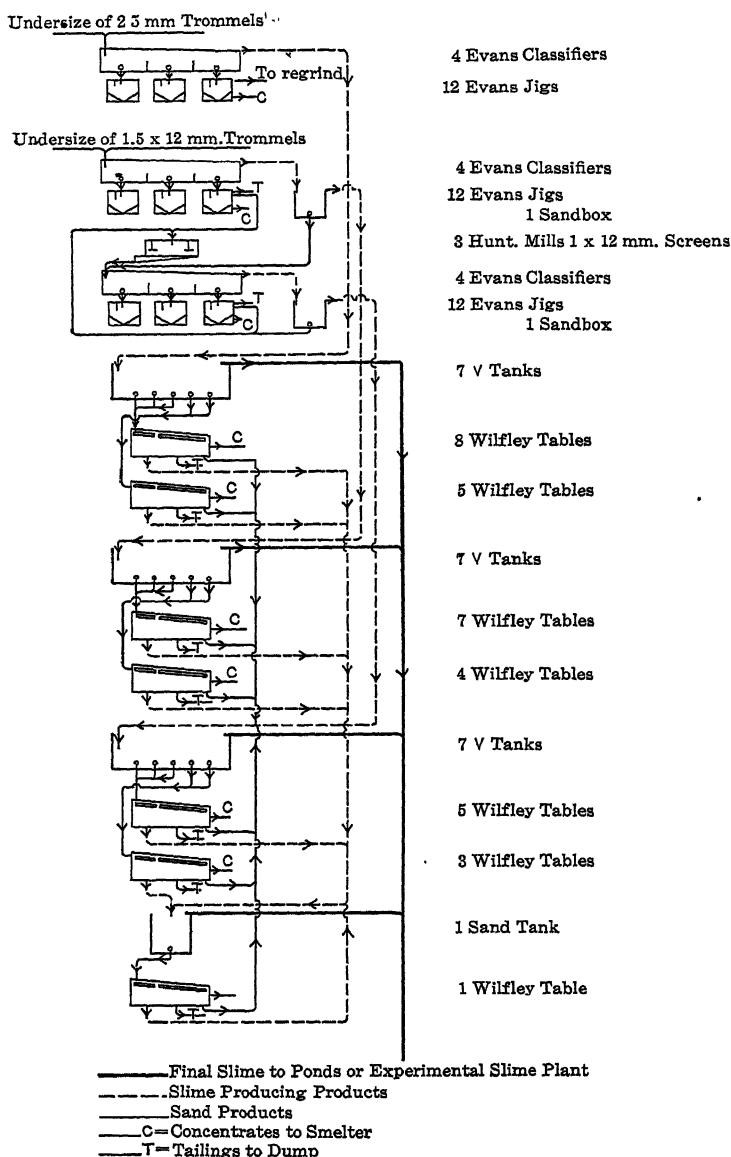
FIG. 2.—SOURCE OF SLIME IN SECTIONS NOS. 2 TO 8.

Sections Nos. 2 to 8 have the original flow sheet. The Evans, launder type, three-spigot classifier is used. There are three sets of four each, and the spigot products are treated on Evans jigs. The

first set, or fine classifiers, receives the undersize of the 2.5-mm. trommels and overflows material which contains the original slime and that produced by the crushers and coarse and fine rolls.   The second set, or middling classifiers, receives the undersize of the 1.5-mm. trommels and overflows material which contains the slime produced in the middling rolls.   The third set, or Huntington mill classifiers, receives the product of the Huntington mills and overflows material which contains the slime produced in the mills.   The last two overflows are desanded in small tanks, the overflows of which, as well as the overflow of the fine classifiers, each go to seven V settling tanks, each 5 by 15 ft., fed at one end and overflowing at the other.   The two sets of spigots, coarse and fine, from the tanks are treated on 32 Wilfley tables, the head waters of which are combined and desanded in a tank about 9 ft. square, having center feed and peripheral overflow.   The plug of this tank is treated on one Wilfley table, the head water of which joins that of the other tables.   The overflow of this square tank plus the overflow of the three sets of V tanks (21 in all) constitutes the slime from Sections Nos. 2 to 8.

Thus the sources of slime in the two flow sheets are similar, and with few exceptions, which will be noted, the slimes are similar.

IV. Constitution of Slimes.

Chemical.—Chemical analyses of slime samples do not vary much. The analyses given in Table I. are of samples covering periods of from several days to a month, and extend over six years.

Table I.—*Chemical Analyses of Anaconda Slimes.*

	Cu. Per Cent.	SiO_2. Per Cent.	FeO. Per Cent.	S. Per Cent.	Al_2O_3. Per Cent.	CaO. Per Cent.	Ag. Oz. Per Ton.	Au. Oz. Per Ton.
June, 1907......	1.83	58.9	3.9	....	19.8	0.4	1.6	0.004
April, 1908......	2.18	61.5	3.7	3.4	18.0	0.3	1.6	0.004
April, 1909......	2.08	60.1	3.5	3.7	21.9	0.2	1.5	0.005
April, 1912......	2.04	58.8	4.5	4.4	19.4	....	....	....
August, 1912......	2.09	62.2	3.7	4.2	18.6	0.8	....	....
September, 1912	2.14	60.8	4.1	4.4	18.9	0.6	2.0	0.005

Attention is called to the percentage of Al_2O_3, which is from 18 to 22 per cent., while the Al_2O_3 in the original ore is only 9 per cent.

Physical.—Physical analyses of the slime show that it consists of granular sand, mineral, and middling grains, and colloid, the whole being in mechanical suspension in water.   The granular materials are fine undecomposed ore constituents.   The colloids are hydrated amorphous substances resulting from the decomposition of some of

the ore constituents, as kaolins, feldspars, etc., and some sulphides. Thus not only is part of the gangue in the colloidal form, but there is some of the copper in the same form, and wet concentration will not recover it. Slime as produced contains approximately 2 per cent. by weight of solids and 98 per cent. by weight of water.

Physical analyses of the slime were made in the following manner:

A sample of slime was procured from the main slime flume, and the colloid separated by repeated dilution and settling out of the sand, followed by a final settling of the colloid, or hydrated amorphous material, by the addition of clear lime water.
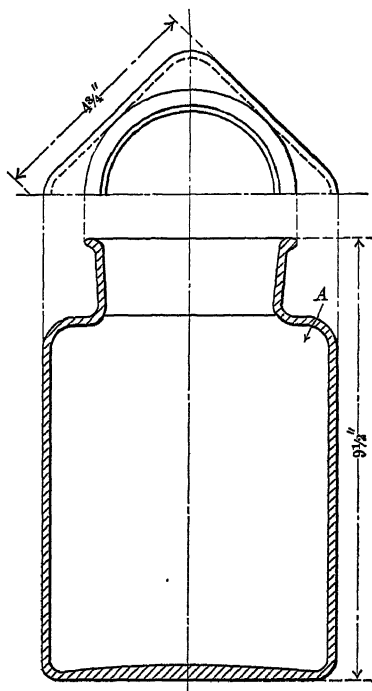
FIG. 3.—GLASS JAR FOR SLIME-SETTLING EXPERIMENTS.

The settling was done in glass candy jars, Fig. 3, which hold about 2.33 liters when filled to the lower end of the neck. These jars are found to be very convenient for such work, as the wide mouth permits rapid pouring, and the shoulder A serves as a trap to prevent the unintentional decantation of fine sand which has already settled.

Thirty-five liters of slime water were divided among a number of these jars, and after standing 15 sec. most of the water was poured into other jars and allowed to stand another 15 sec.; this operation being repeated till but very little additional sand settled out. The

residues, containing the coarser sand, were all combined and re-treated in the same way to obtain a sand product nearly free from colloid. The decanted water was then diluted to about twice the original volume, which reduced the size of the individual masses of coagulated, amorphous colloid, and thus lessened their rate of settling, so that, on standing 30 sec., considerable sand settled out, and but very little colloid. This operation was also repeated several times, and the approximately clean sand combined with the coarser sand. The decanted water was again diluted and allowed to stand for still longer periods, this process of stage dilution being continued till the original 35 liters
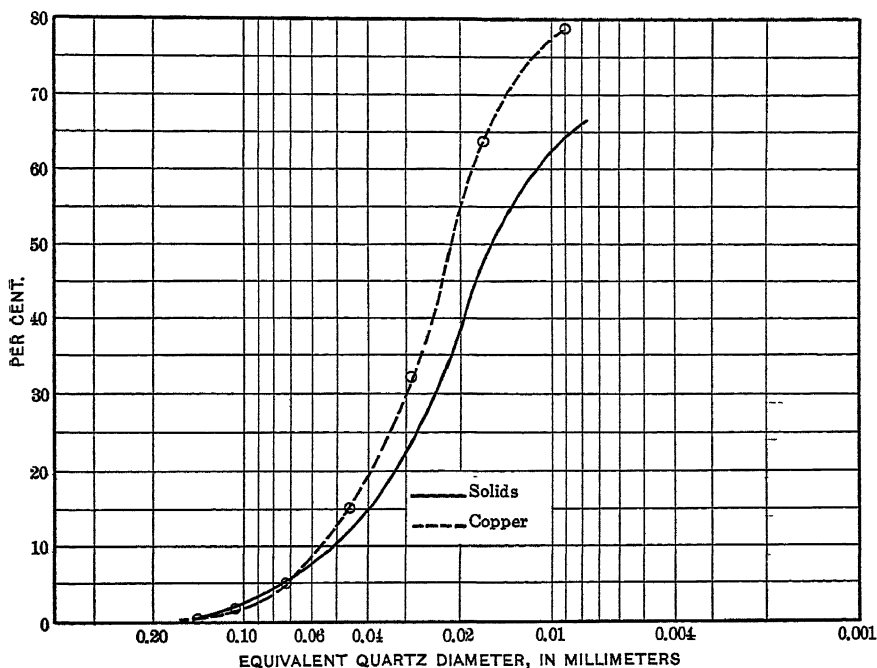
FIG. 4.—SETTLING TESTS OF GENERAL CONCENTRATOR SLIMES.

had become about 800 liters. The very finest crystalline material settled out of the highly diluted colloid in 45 min., but further standing of the decanted colloid for 1.5 hr. failed to settle any more sand. Considerable of this sand-free colloid remained in suspension for more than 24 hr., but was finally precipitated by the addition of a moderate amount of clear lime water.

The approximately clean sand was freed of the last of the colloid, by highly diluting with clear water, and allowing to settle an hour or more, decanting, and repeating the washing till no more colloid

remained, all these wash waters being added to the main bulk of sand-freed colloid.

The clean sand was classified by free settling in the same glass jars, first for 960 sec., pouring off the top 6 in. of water, filling the jar again, and repeating four times. The settled sand was similarly treated for successive periods of 480, 240, 120, 60, 30, and 15 sec. Each product was dried, weighed, and assayed. To determine the range of sizes of the sand products, microscopic measurements were
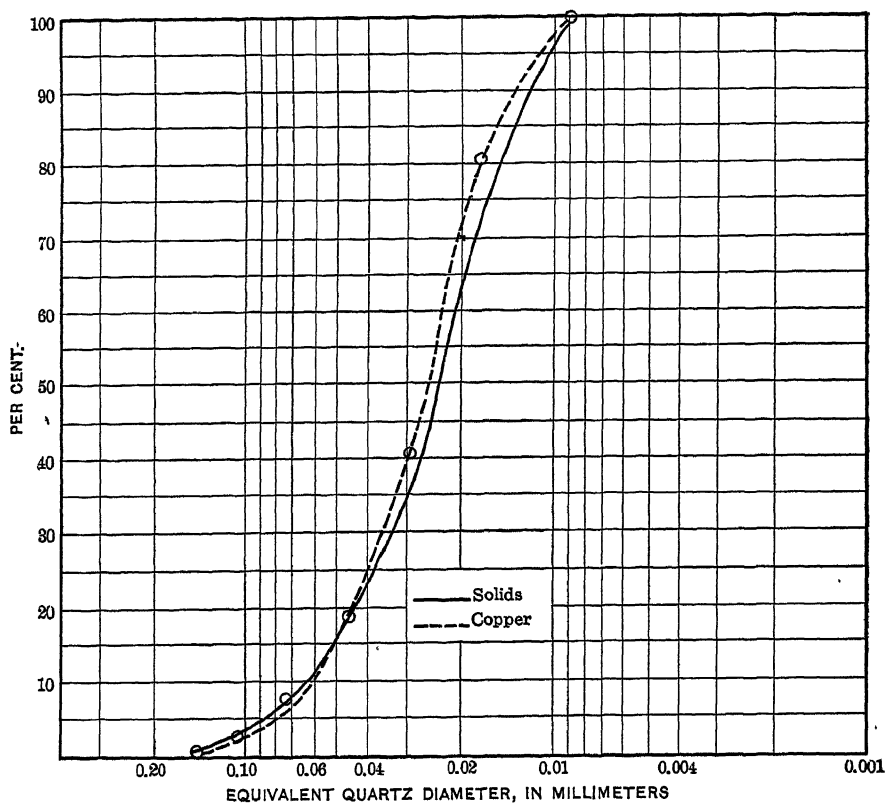
FIG. 5.—SETTLING TEST OF SAND PORTION OF SLIMES.

made on clean quartz obtained under the same settling conditions as those of the sands. From this data, curves, Figs. 4 and 5, were plotted, the former being the settlement curve for the general slime and the latter that for the sand portion only.

During the classification test a small amount of " Secondary colloid " was produced, owing to the chemical effect of water on the large area exposed by the finely divided sands.[2]

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[2] Cushman: *Bulletin* Nos. 85 and 92, *Bureau of Chemistry, U. S. Department of Agriculture.*

Table II gives the results of such a physical analysis made on a sample of mill slime; and Table III gives chemical analysis of the colloidal portions.

From Fig. 5, 96 per cent. of the slime is finer than 200 mesh (0.08 mm.).

TABLE II.—*Settling Test on Anaconda Slime.*

Velocity of Settling Per Second.	Diameter of Quartz of this Settling Velocity.	Solids		Copper.		
		Direct Per Cent.	Cumulative. Per Cent.	Assay.	Direct Per Cent.	Cumulative Per Cent.
Millimeters.	Millimeters.					
10.16 +	0.146 +	0.6	0.6	1.40	0.4	0.
10.16 — 5.08	0.146 — 0.106	1.2	1.8	1.96	1.1	1.
5.08 — 2.54	0.106 — 0.074	3.1	4.9	2.54	3.5	5.
2.54 — 1.27	0.074 — 0.046	7.3	12.2	3.05	10.0	15.
1.27 — 0.635	0.046 — 0.029	11.1	23.3	3.46	17.2	32.
0.635 — 0.318	0.029 — 0.017	24.0	47.3	2 94	31.4	63.
0.318 — 0.159	0.017 — 0.009	17.1	64.4	1.96	15.0	78.
0.159 — 0.000	0.009 —	0.4	64.8	0.86	0.1	78.
Total sand.............	....	64.8	....	2.72	78.7	....
Primary colloid.....	....	33.0	97.8	1.47	20.3	99.0
Secondary colloid...	....	2.2	100.0	1.13	1.0	100.0
		100.0	....	2.27	100.0	....

TABLE III.—*Chemical Analysis of Colloids.*

	Free Water.	Ignition Loss.	SiO_2.	Fe.	S.	Al_2O_3.	CaO.	Cu.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Ct.
Primary colloid......	0.46	8.08	52.5	1.7	2.3	24.8	0.9	1.47
Secondary colloid....	0.43	5.18	58.1	1.3	1.7	25.4 •	0.9	1.13

V. THE SLIME PROBLEM.

The problem of wet concentration of the Anaconda slimes may be divided into two parts, viz.: First, to prepare the slime for use on some concentrating machine by thickening it to the proper density, and at the same time recover water clean enough for re-use in the mill; and second, to recover the mineral values from the thickened pulp in a concentrate which can be smelted.

The quantity of material to be handled calls for an extensive plant and consequently the most efficient and economical installation is sought. The quantity of slime produced is given in Table IV.

TABLE IV.—*Quantity of Slime Produced.*

Ore Treated Per Section Per 24 Hr.		Assay.	Slime Produced Per Section Per 24 hr.		Assay.	Density.	Per Cent. of Original.	
							Ore.	Copper.
	Tons	Per Cent. Cu.	Gallons.	Tons	Per Cent. Cu	Per Cent.		
Section 1	1,470	3.25	2,275,700	193.6	1.85	1.98	12.9	7.4
Sections 2 to 8..	1,313	3 24	2,646,000	225.0	2.03	2.01	17.2	10.8

The present output of the mill, based on 10,000 tons of ore treated per 24 hr., is 18,401,000 gal. of pulp containing 1,660 tons of dry slime.

The first part of the problem, then, is to thicken 18,401,000 gal. of pulp at 2 per cent. density to a pulp of 10 per cent. density, necessitating the removal of 15,700,000 gal. of water. The second part is to obtain machines of suitable capacity, efficiency, and cost of maintenance to treat 1,660 tons of dry slime and produce a concentrate which can be smelted.

VI. EXPERIMENTAL SLIME PLANT.

All of the slime produced previous to July, 1912, was sent to the slime ponds, located below the works, and whatever settled in them was excavated and stored in a pile adjacent to the ponds. A limited amount of the excavated material has been smelted in the blast furnaces. Since the summer of 1911 the Peck centrifugal concentrator has been treating a portion of the slime settled in one of the ponds.

Thus there had been little done to treat all of the slime direct from the mill until July, 1912, when Section No. 1 of the concentrator was remodeled. No decision had been reached as to the best method for treating the slime: but from data obtained from slime experiments conducted at Great Falls, sufficient Callow tanks and round tables were installed in an experimental slime plant to treat one-fourth of the slime produced in one section of the mill. This flow sheet is given in Fig. 1. Since the installation several thickening and concentrating devices have been added and competitive tests made upon them, to determine the machines best adapted to the solution of the slime problem. The various thickeners include the Kuchs-Laist centrifugal separator, the Callow tank, the Garred filter, and the Dorr continuous thickener. The concentrating machines include the round table and four standard makes of concentrators, the latest slime machines on the market. All these machines have received slime from Section No. 1, slime from Sections Nos. 2 to 8, and the total classifier overflow of Section No. 1.

VII. THICKENERS.

The simplest slime thickeners are the open rectangular tanks and the slime ponds, the latter being enlargements of the former. A modification of the open tank is the Callow tank, which has center feed and peripheral overflow. Mechanical thickeners include the Kuchs-Laist centrifugal separator, the Garred filter, and the Dorr continuous thickener. A brief description of the several devices and the results of tests made upon them are given below.

Slime Ponds.—There are six of these, each 300 ft. wide and 600 ft. long, and average 15.5 ft. deep. These are operated in cycles, and when possible three are used in series. As soon as a pond becomes filled, it is allowed to drain, and the resulting settlement, containing 35 per cent. of moisture, is excavated by a Lidgerwood bucket excavator. No very complete data of the operation of the ponds are available, but the degree of settlement has been governed by the extent to which excavation has been carried on and empty ponds provided. Whatever has been excavated has been stored for future treatment, by smelting or otherwise.

One of these ponds has been reserved to thicken slime for the Peck machine. It was hoped that the pond could be partly filled with slime and then, by pumping from different depths, a pulp obtained of the proper density for feed : viz., 20 per cent. of solids by weight. Several attempts were made to perfect this scheme, but with no success. The scheme finally adopted is as follows: Two large tanks, 16 by 16 ft. and 8 ft. deep, supplied with agitators, are located under a hopper-bottomed bin, into which is dumped slime dredged from the pond. A portion of this is charged into one of the agitation tanks and mixed with water pumped from near the surface of the pond, until a pulp of about 20 per cent. density is obtained. This is fed to the Peck machine. Meanwhile the other tank is preparing a charge, which is held in readiness until the first tank is emptied. In this way a uniform feed is obtained. This, however, is rather a crude way, particularly where large tonnages are to be handled.

If the ponds are fed with the proper quantity of slime complete settlement is obtained; but difficulty arises in recovering the slime at the proper density.

Tanks.—Open tanks include any tanks where no baffles are used, and free settling is used. They may be fed at one end and overflow at the other, as in rectangular tanks, or they may be fed at one or more interior points and overflow peripherally, as in square, round, or rectangular tanks. Baffle tanks have some arrangement of baffles (either vertical or sloping) and may be fed the same as the plain tanks.

A tank 10 ft. 10 in. by 9 ft. 6 in. and 8 ft. deep, fed at the center, having peripheral overflow, and having one spigot from the center of the bottom, gave the following results when fed with slime at 2.5 per cent. density. In this tank the slime forms its own hopper in the bottom of the tank. The spigot density was about 8 per cent.

Gallons Per Minute.		Per Cent of Solids
Total.	Per Square Foot.	In Spigot.
20	0.194	100.0
22	0.213	95.0
25	0.233	90.0
28	0.272	85.0
31	0.301	80.0
34	0.330	75.0
38	0.369	70.0
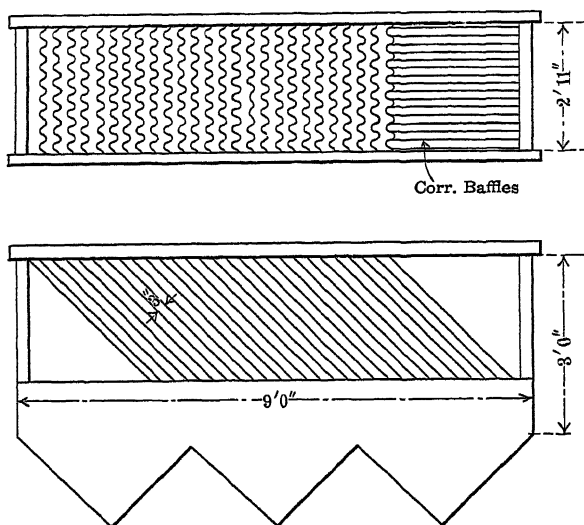
FIG. 6.—EXPERIMENTAL BAFFLE TANK.

Comparative Efficiency of Open and Baffle Tanks:—For this experiment, a tank 3 ft. wide, 9 ft. long, and 3 ft. deep, with three hoppers, was used, Fig. 6. Feed entered at one end over an apron, so as to produce as little disturbance as possible, and overflowed at the opposite end, over the entire width. Corrugated iron baffles sloping 45° were placed in different positions, including longitudinal and transverse, and at different depths below the surface. The results of the various tests are given in Table V. and plotted in Fig. 7, the latter also showing a composite curve, *A*, of all the baffle tests. Table VI. summarizes the tests under varying quantities of feed. From this table, if perfect settling is required a baffle tank has no advantage, but if it is desired to slough off a portion of the solids, the baffle tank has a greater capacity.
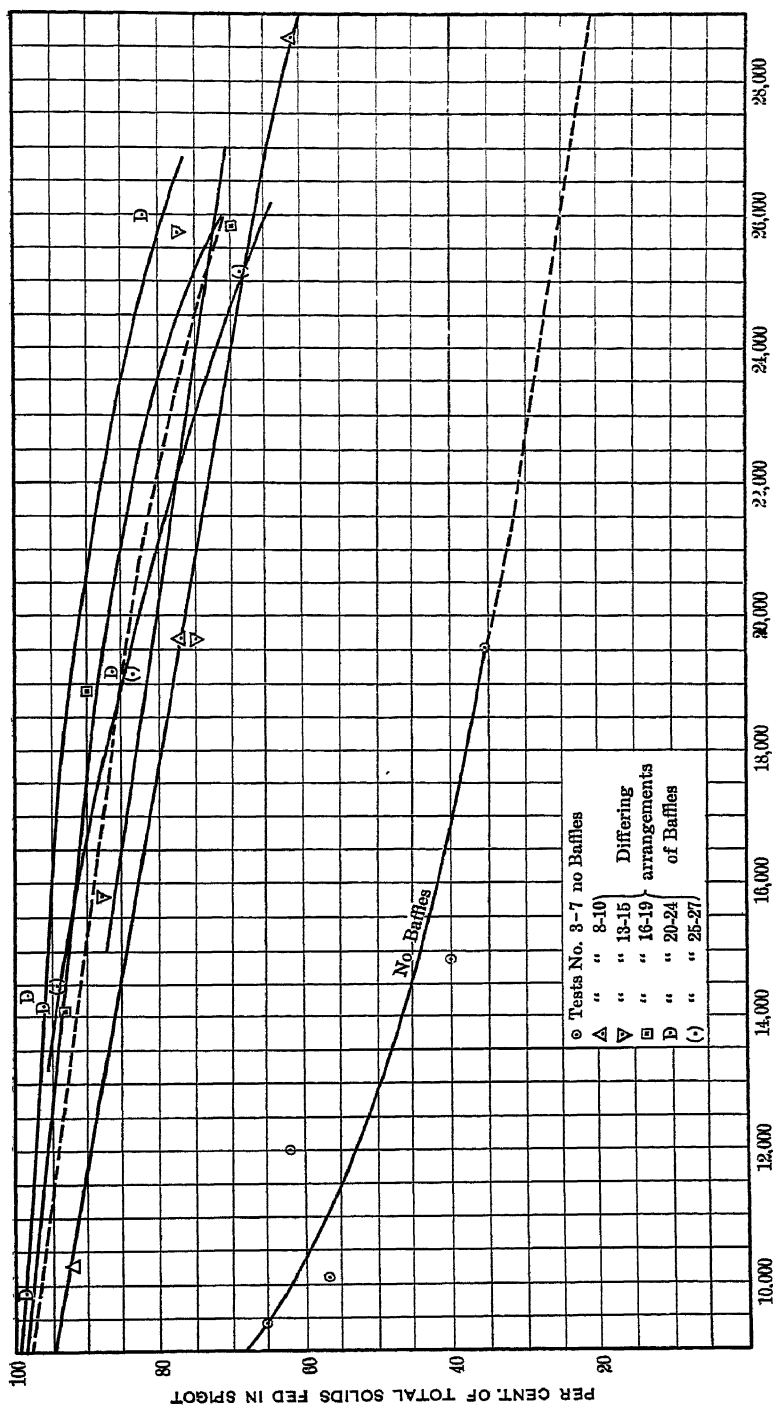
FIG. 7.—CURVE OF BAFFLE TANK DATA.

TABLE V.—*Slime Settling in Open and Baffle Tanks.*

Test No.	Arrangement of Baffles	Feed to Tank.		Feed to Tank Per Foot of Width.		Spigot Discharge from Tank.			
		Gallons Per 24 Hr.	Pounds Solid Per 24 Hr.	Gallons Per 24 Hr.	Pounds Solid Per 24 Hr.	Per Cent. of Total Vol'me.	Density Per Cent. of Solid.	Per Cent. of Total Solids Fed	Per Cent. of Total Copper Fed.
3	No baffles...	9,360	1,300	3,120	433	9.4	10.9	65 9	68.1
4	Ditto	10,075	1,545	3,358	515	9.4	10.4	57.4	59.4
5	Ditto	12,010	1,655	4,003	552	7.6	12.6	63.1	66.3
6	Ditto	14,900	2,290	4,966	763	5.4	12.2	39.7	41.2
7	Ditto...	19,500	3,240	6,500	1,080	4.6	13 8	35.3	37.4
8	Longitudinal baffles 4 in. deep a	10,250	1,540	3,416	513	12.8	11.9	92.3	93.4
9	Ditto..	19,715	2,960	6,572	987	8.0	15 7	76.9	79.0
10	Ditto..	28,630	4,300	9,543	1,433	4.6	20.5	61.1	63.4
13	Longitudinal baffles 0.5 in deep	15,905	2,390	5,302	797	8.3	17.0	87.6	88.7
14	Ditto.....	19,704	2,965	6,568	988	4.7	24.3	74 9	77.6
15	Ditto	25,885	3,885	8,628	1,295	4.5	25.4	76 6	79.4
16	Crosswise baffles 4 5 in. deep a with tops sloping away from feed end of tank . .	9,955	1,493	3,318	498	11.2	14.2	97.4	97.8
17		14,125	2,120	4,708	706	7.9	18.7	98.6	94 5
18		18,965	2,845	6,322	948	7.4	18.8	89.4	91.2
19		25,855	3,880	8,618	1,293	4.2	25.5	70.6	75.6
20	Ditto, 0 5 in, a	9,785	1,460	3,245	487	9.8	16 2	98.5	98.8
21	Ditto.........	14,360	2,155	4,787	718	8.7	17.7	98.3	98.8
22	Ditto....	14,255	2,140	4,752	713	7.5	20.0	96 0	96.8
23	Ditto....	19,070	2,865	6,357	953	5.4	24.2	86.8	90.9
24	Ditto..	25,965	3,900	8,655	1,300	4.5	26.9	82 3	85.9
25	Crosswise baffles 0.5 in deep a with tops sloping toward feed end of tank....	14,355	2,155	4,785	718	4.8	28.6	93.4	94.8
26		19,040	2,860	6,347	953	6.0	21.7	84.8	87.5
27		25,010	3,755	8,337	1,252	3.6	27 7	68.3	72.2
28	Ditto, but 1 5 in. deep a at feed end and 4 in. at overflow end of tank.	19,065	2,865	6,358	955	5.1	22.1	76.7	80.1
29		25,475	3,825	8,492	1,275	4.1	23.4	62 9	63.2

a The depth here, means the depth of the top edges of the baffles below the overflow of the tank.

TABLE VI.—*Summary of Open and Baffle Tank Data.*

Gallons of Feed Per Foot of Width Per 24 hr.	Per Cent. Settled.		Rates of Settling Efficiency.
	Baffle Tank.	Open Tank.	
	Per Cent.	Per Cent.	
1,883a	100.0	100.0	1.00
2,000a	99.9	94.8	1.05
3,000	97.5	67.8	1.44
4,000	94.0	53.5	1.76
5,000	90.3	44.8	2.02
6,000	86.8	38.0	2.28
7,000	82.0	32.6	2.52
8,000	75.9	28.0	2.71
9,000	67.8	23.8	2.85

a These figures estimated by extending the curves.

Callow Tanks.—The Callow tanks in the slime plants are 8 ft. in diameter, center feed, peripheral overflow, and discharge through goose necks from the bottom of the cone. Twenty of these receive the classifier overflow. They are each handling 118,700 gal. in 24 hr. (82.4 gal. per minute) of pulp at 2 per cent. density and discharging 40 to 45 per cent. of the solids in a pulp of 8 per cent. density. Twenty-four tanks are used to thicken whatever feed is to be treated

on the round tables or competing machines. At the Great Falls plant tanks corresponding to these are handling 25 gal. per minute and overflowing clear water, but owing to the difference in constitution of the slimes 9 to 11 gal. per minute is the maximum capacity when settling Anaconda slime to a clear water overflow. The operation of these tanks under different conditions of feed and efficiency is shown in Table VII.

TABLE VII.—*Operation of Callow Tanks Under Different Feed Conditions.*

A. Feed : Total classifier overflow from Section No. 1.

	Gallons Per Tank Per Minute.	Pounds of Solids Per Tank Per Minute.	Pulp.		Per Cent. of Total.	
			Density.	Assay. Per Cent. Cu.	Solids.	Pulp.
Feed..........	11.1	1.93	2.1	2.22	100.0	100.0
Spigot.......	1.9	1.85	11.0	2.26	94.5	95.5
Overflow....	9.2	0.10	0.1	1.56	5.5	4.5

B. Feed : Slime from Section No. 1.

Feed.........	12.1	1.73	2.0	1.83	100.0	100.0
Spigot.......	1.9	1.64	10.0	1.85	94.5	95.4
Overflow...	10.2	0.09	0.1	1.56	5.5	4.6

C. Feed : Same as *B.*

Feed.........	11.6	1.63	2.0	1.84	100.0	100.0
Spigot......	1.9	1.48	9.0	1.88	91.1	93.0
Overflow...	9.7	0.15	0.2	1.43	8.9	7.0

D. Feed : Slime from Sections No. 2 to 8.

Feed.........	13.9	2.37	2.0	2.03	100.0	100.0
Spigot.......	1.6	2.11	14.5	2.07	89.2	91.2
Overflow...	12.3	0.26	0.3	1.66	10.8	8.8

E. Feed : Same as *D.*

Feed..........	10.9	1.91	2.0	1.93	100.0	100.0
Spigot......	1.7	1.79	11.7	1.98	94.0	95.6
Overflow...	9.2	0.12	0.1	1.45	6.0	4.4

Kuchs–Laist Centrifugal Separator.—This machine is designed to remove the colloidal material and excess water from the slimes and produce a spigot discharge containing all the granular material and which is suitable for feed to some slime-concentrating machine. As shown in Fig. 8, it consists of a set of radial chambers mounted on a horizontal hollow shaft, about which it revolves. At the outer periphery of each chamber is a spigot discharge plug to remove the heavier material which settles in the chambers due to centrifugal force. Feed enters at one end of the hollow shaft, through a stuffing-box connection to the feed pipe, and passes to the chambers, which are provided with baffles. These baffles serve to check the

4' 5"

1½" Pipe

3" Discharge Pipe

Concrete Foundation

CROSS SECTION A-B

Overflow

Hollow Shaft

From dotted line out
⅜ ¼" holes spaced to
cen. line to cen. line

Spigot Product

Plug

Baffles
spaced
2" c.-c.

A

B

2' 5½"

Feed Entry
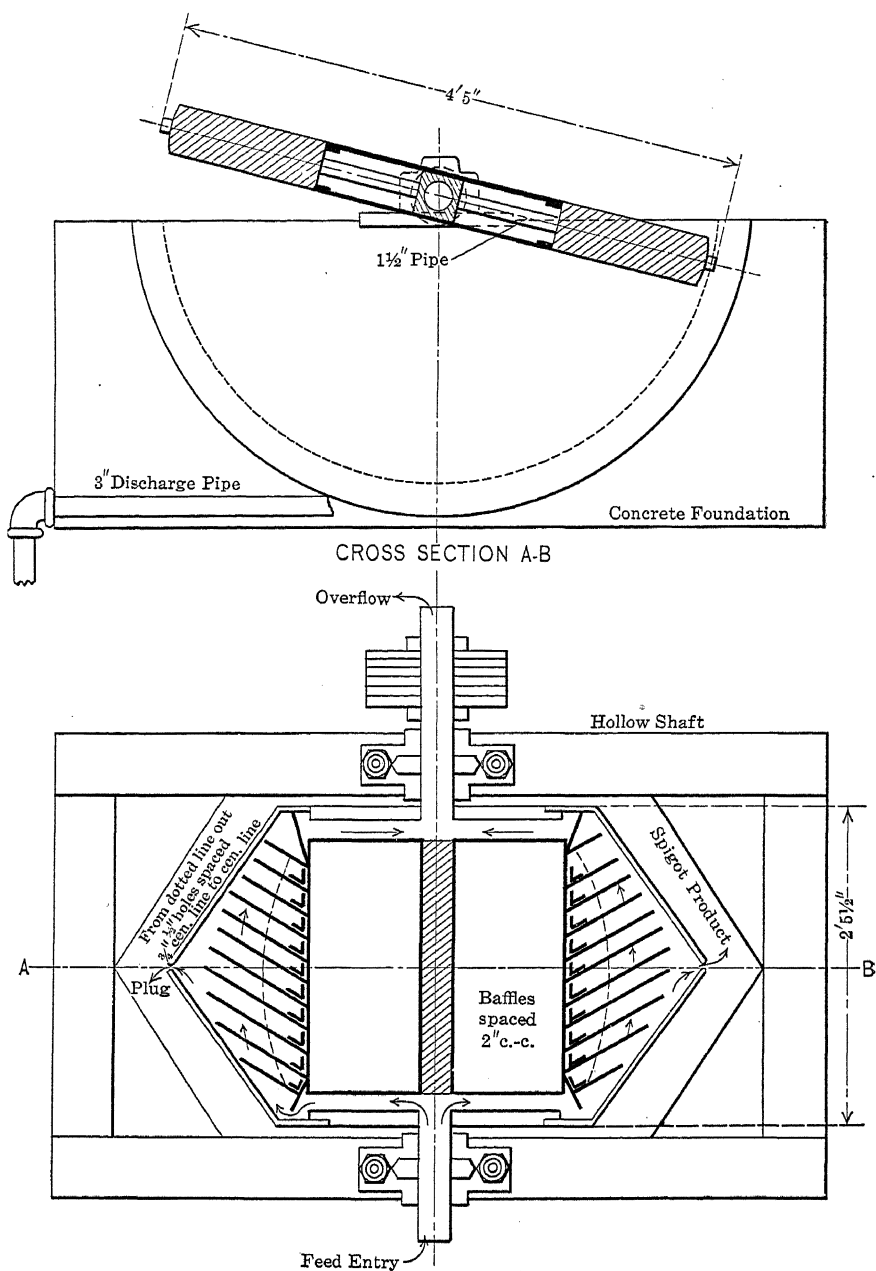
FIG. 8.—KUCHS-LAIST CENTRIFUGAL SEPARATOR.

flow of material and direct the heavier material to the point of discharge. The heavier material is discharged through the plugs, and the lighter is forced back to the hollow shaft and is discharged at the opposite end from which it enters. This particular machine has two chambers, but the circumference could be completed with chambers so that the machine would have about 12 times its present capacity. The speed has been varied from 640 to 1,200 rev. per min., and, roughly, the capacity varies directly with the speed, if the same density of spigot product be maintained.

Considerable difficulty was experienced in obtaining material of which to make discharge plugs which would wear a reasonable length of time. Plugs of cast iron, steel, topaz, agate, and rubber were used. The holes in these plugs varied from $\frac{1}{16}$ to $\frac{1}{8}$ in. when new and gave a spigot density of from 15 to 30 per cent. After running from 3 to 5 hr. these holes were enlarged to such an extent that the density dropped to from 8 to 15 per cent., and in the case of rubber plugs the holes became choked with pieces of rubber torn from the plug. Finally a set of plugs of sintered alumina (called "alundum") having a hole $\frac{1}{30}$ in. in diameter was tried. These plugs were in use for 85 hr. and the density dropped from 30.7 to 16.2. The high density at the start is due to the small opening.

Between the speed limits of 640 and 1,200 rev. per min. the speed at which the machine is run does not materially affect the life of the plug.

Table VIII. gives the data of two tests run at 1,200 and 640 rev. per min., respectively.

TABLE VIII.—*Operation of Kuchs-Laist Centrifugal Separator on Slime from Sections Nos. 2 to 8.*

A. Speed of machine, 1,200 rev. per min.

	Gallons.		Solids.		Copper.		Density.
	Per Minute.	Per Cent. of Total.	Pounds Per Minute.	Per Cent. of Total.	Assay.	Per Cent. of Total.	
Spigot.........	8.57	9.3	10.2	61.7	2.90	76.1	13.1
Overflow......	83.76	90.7	6.3	38.3	1.47	23.9	0.9
Total.......	92.33	100.0	16.5	100.0	2.35	100.0	2.1

B. Speed of machine, 640 rev. per min.

	Per Minute.	Per Cent. of Total.	Pounds Per Minute.	Per Cent. of Total.	Assay.	Per Cent. of Total.	Density.
Spigot.........	5.43	9.8	5.0	55.1	2.36	67.9	10.3
Overflow......	50.82	90.2	4.1	44.9	1 37	32.1	0.7
Total.......	56.25	100.0	9.1	100.0	1.92	100.0	2.0

To show the extent to which the colloid is separated, a colloid analysis, as previously described, was made on the product of a test similar to *B*, Table VIII., with the results shown in Table IX.

TABLE IX.—*Colloid Analysis of Products of Kuchs-Laist Separator.*

	Spigot.					Overflow.					Total Products.			
	Solids			Copper.		Solids			Copper.		Solids		Copper.	
	Per Cent. of Total Spigot.	Per Cent. of Feed.	Assay	Per Cent of Total Spigot	Per Cent of Feed.	Per Cent of Total Overflow	Per Cent. of Feed.	Assay	Per Cent. of Total Overflow.	Per Cent. of Feed.	Per Cent. of Total	Assay	Per Cent. of Total	Per Cent. of Total.
Sand...	85.0	56.1	2 96	91.6	76.8	7 2	2.5	1.96	14 1	2.3	58.6	2.92	79.1
Colloid	15.0	9.9	1 54	8.4	7.1	92.8	31.5	0.95	85.9	13.8	41.4	1.09	20.9
Total ..	100 0	66.0	2 75	100.0	83 9	100.0	34.0	1.02	100.0	16.1	100.0	2.16	100.0

Garred Filter.—An experimental Garred filter has recently been installed to prepare feed for the round tables, slime from Sections Nos. 2 to 8 being used for feed. The filter consists of 100 leaves, each 6 by 10 ft., made of 10-oz. canvas, and connected to a centrifugal pump. The 100 leaves are divided into five sections of 20 leaves each, and connected with water at 15 lb. pressure in such a way that the suction on the several sections can be cut off and the water turned into the leaves. After the filter has been in operation for a while, the suction is cut off from one section of leaves and the clean water turned on. This forces the cake, which is formed on the outside of the leaf, from the leaf, and it falls to the bottom of the tank containing the leaves. The several sections are similarly treated, with the result that a thickened pulp is discharged through plugs provided in the bottom of the tank.

The filter has the advantage of positive action, with 100 per cent. efficiency in settling, provided the leaves are kept in good condition. This filter treating slime at the rate of 400 gal. per minute produced a pulp of 11.9 per cent. density. The filter·was run at this capacity, as it produced sufficient pulp to feed the eight round table decks used to treat the pulp. No definite data as to maximum capacity has been obtained, but in all probability the 100-leaf filter will handle 1,000 gal. of slime per minute as an average during the life of the filter leaves.

Dorr Continuous Thickener.—One Dorr thickener was installed near the slime plant, and fed with slime from Sections Nos. 2 to 8. As this was just recently installed, only a few tests have been made upon it, but the results are very favorable.

The tank is cylindrical, is 27.5 ft. in diameter and 9.7 ft. deep, and has center feed and peripheral overflow. The discharge is through one spigot at the center of the bottom. Near the bottom of the tank is a set of four rakes attached to a central vertical shaft which makes about 3.3 rev. per hour, and rakes the settled slime to the point of discharge. The rakes are set at 12° with the horizontal, so that a flat cone of the slime is formed. In later tests the rake arms were made horizontal. This machine has the advantage of, first, a large settling area, without the disadvantage of the conical tank with subsequent excessive loss of mill head; and, second, only one spigot of reasonable size, instead of numerous small spigots as in the case of small settling units. The capacity of the 27.5 ft. Dorr thickener per square foot of settling area is practically the same as that of the Callow tanks when operated to the same efficiency.

Experiments were made to determine the capacity and efficiency of different depths of tank. For this purpose the overflow launder was lowered to make the tank 3 ft., and then 2 ft. deep. The rakes were made horizontal. The data obtained are given in Table X., and indicate that when operating to about 98 per cent. efficiency a 3-ft. tank has 85 per cent. of the capacity of a 9.7-ft. tank; while a 2-ft. tank has about 85 per cent. of the capacity of a 3-ft. tank.

TABLE X.—*Treatment of Slime from Sections Nos. 2 to 8 in Dorr Continuous Thickener.*

Depth of Tank	Feed.			Spigot Discharge.					Overflow.			
	Rate Per 24 Hr.		Density. PerCent.	Rate Per 24 Hr.		Density. PerCent.	Per Cent of Total.		Rate Per 24 Hr.		Per Cent. of Total.	
	Pulp.	Solids.		Pulp.	Solids.		Pulp.	Solids.	Pulp.	Solids.	Pulp.	Solids.
Feet.	Gal.	Lb.	Solids.	Gal.	Lb.	Solids.			Gal.	Lb.		
9.7	195,110	34,380	1.90	39,860	34,205	9.6	20.0	99.5	155,250	175	80.0	0.5
3.0	163,350	31,250	2.26	24,210	30,640	13.2	14.9	98.0	139,140	610	85.1	2.0
2 0	162,940	29,820	2.13	22,800	26,880	12.7	14.0	90.	140,140	2,940	86.0	10 0

VIII. CONCENTRATORS.

Two types of concentrating machines have been tested very thoroughly, both on short and long time tests. These are the round table and the Peck centrifugal concentrator. In addition to these, competitive tests were made on four standard machines, each furnished by a different manufacturer, and which are the latest types of slime machines on the market. As these machines were furnished by the makers and there were differences in the recoveries made by the several machines, and not wishing to publish data detrimental to the less efficient ones, when treating this slime, the results of these tests will be discussed only in a general way.

Round Table.—Probably one of the oldest types of concentrating machines, the round table, or "buddle," has been revived of late years, and with the improvements made in the construction, is now a serviceable and efficient machine. Those installed in the experimental slime plant and on which these tests were made have a steel frame, with concentrating surfaces of cement. They are 18 ft. in diameter, and are built in stands of four decks each, the decks being spaced about 5 ft. apart. The whole stand is driven from the top and makes 12 rev. per hour.

In practically all the work, each of the eight decks has received the same kind of feed, and any middling product resulting from them has been returned to the feed of the decks. Feed has been prepared principally by Callow tanks, but the Garred filter and the Kuchs-Laist separator have been used to some extent. The first two thickeners in combination with the round tables give about the same recovery, but with the latter the recovery is slightly higher, as the separator removes a great deal of the colloid, and gives a cleaner sand product to the round table.

Table XI. gives the results of treating the products of the several thickeners on the round tables, and the effect of removing the colloids is evidenced by the part *C*, even when figured back to the original slime.

TABLE XI.—*Operation of Round Tables.*

A. Preceded by Callow tanks.

	Solids. Pounds per 24 Hr. per Deck.	Assay.			Copper. Pounds per 24 Hr.	Per Cent. of Total.	
		Cu.	Insol.	FeO.		Solids.	Copper.
		Per Cent.	Per Cent.	Per Cent.			
Feed	9,325	2.18	....	....	204	100.0	100.0
Concentrate........	1,425	7.31	55.1	13.8	104	15.3	51.2
Tailing.............	7,250	1.21	....	....	88	77.8	43.1
Tank overflow...	650	1.81	....	....	12	6.9	5.7

B. Preceded by Garred filter.

	Rate per Deck per 24 Hr.		Density. Solids.	Assay.		Copper. Pounds per 24 Hr.	Per Cent. of Total.	
	Gal.	Lb.		Cu.	Insol.		Solids.	Copper.
			Per Cent.	Per Cent	Per Cent.			
Feed.............	30,625	11,199	11.9	1.98	....	222	100.0	100.0
Concentrate......	13,803	1,631	....	7.02	58.8	114	14.6	51.6
Tailing...........	16,822	9,568	....	1.13	....	108	85.4	48.4

C. Preceded by Kuchs-Laist separator.

Test No.	Separator Spigot Round Table Feed.						Round Table Concentrates.					
	Rate per 24 Hr.		Density Per Cent.	Assay. Per Cent.	Per Cent of Original.		Per Cent. of Original.		Per Cent. of Cu in Feed to Table.	Assay.		
	Gal.	Lb.	Solids.	Cu.	Solids.	Copper.	Solids.	Copper.		Cu.	Insol.	FeO.
										Per Cent.	Per Cent.	Per Cent.
1	8,240	12,260	16.0	2 26	67.0	78.2	16.0	55.9	71.5	6.75	59.6	13.0
2	8,380	11,320	14.5	2.46	68.6	78.5	16.7	58 5	74.5	6.95	57.7	13.7
3	9,050	11,300	13.0	2.53	65.9	78.6	14.8	53.6	68.2	7.69	52.5	14.9
4	9,800	13,070	14.0	2.81	61.0	76.9	13.3	56.0	72.8	9.68	47.1	16.8

One round table deck used in this test.

Peck Centrifugal Concentrator.—The Peck machines, of which there are two, have been treating the slime which has settled in one of the slime ponds, which under the present operating conditions are sloughing off an appreciable amount of colloid and fine sand. No direct comparison of the work of this machine with that of the round table can be made except where the feed for the two machines is prepared in a similar manner.

The Peck concentrator is an intermittent centrifugal machine, and consists primarily of two cones, one within the other, both revolving in the same direction about a vertical shaft. The feed is introduced through an annular feed box, attached to the shaft, into the annular space between the two cones at the lowest point, and rises through it, where the concentration takes place. Centrifugal force throws the solids to the inner wall of the outer cone, and the water in the feed, rising in the annular space, carries the lighter material along with it and is discharged through a set of plugs distributed around the upper edge of the outer cone. The result is, a layer of concentrate adheres to the wall of the cone and the tailing has been discharged through the plugs. When this layer is sufficiently thick the feed is shut off; the outer cone is slowed down, thus reducing the centrifugal force on the concentrate; and wash water is turned in. At the same time launder connections with the discharge plugs are changed, so that a middling product is discharged, and finally the concentrate itself is discharged.

The effect of centrifugal force is to increase the apparent weight of the slime constituents and thus allow of greater capacity, but there is, of course, no effect on the ratio of the specific gravities of these constituents.

The changes of the cycles, which are about 3.5 min., are accomplished by an ingenious automatic device, operated hydraulically.

The older of the two machines treats 120 tons of dry slime per day. The speeds of its cones are 473 rev. per min. for the outer cone and 271 for the inner cone. The corresponding figures for the newer machine are 150 tons, and 505 and 285 rev. per min.

Competitive Tests on Standard Concentrating Machines.—These four machines were fed with slime from Sections Nos. 2 to 8, slime from Section No. 1, and the total classifier overflow from Section No. 1, and the results were very consistent. Tests on any one feed were run simultaneously on all the machines, the feed being distributed by a rotary feeder to be described. The results of these tests compared with the work of the round table under operating conditions show that the best machine does slightly better work than the round table; that the next best machine does just about the same work; but that the other two are not as efficient. Assuming that the machines were doing their best work when being tested, under the supervision of representatives of the manufacturers, the round table would be just about as efficient as the best of them if they were under operating conditions.

Twenty-Deck Round Table.—As the round table type of concentrator has proved so satisfactory as regards floor space occupied, maintenance, operation, tonnage, and efficiency, a machine of this type with 20 concentrating surfaces superimposed one on the other has been constructed, Fig. 9. This may be called a multiple-deck concentrator, the 20-deck machine being a special case. This 20-deck round table occupies a circle of floor space 20 ft. in diameter and is 30 ft. high including the feed box. It will therefore occupy about the same space as the four-deck stands previously installed, but will have five times the capacity.

The decks are about 1 ft. apart, allowing room for inspection of each deck. The bottom deck acts as a support for the upper ones, and is itself supported near the periphery by wheels running on a circular track. The power is applied by a motor, through gears, to a circular rack attached to the lower deck. The surfaces of the decks are concrete made of two parts tailing sand which is under 1.5 mm. and one part cement, and are supported by sheet steel. The concrete is reinforced with expanded metal lathing fastened to the sheet metal. The rest of the machine, other than the concentrating surfaces, is of steel. Suitable steel launders are provided on each deck for the various products of the decks. Each deck also has its own feed and wash water box.

In the center of the structure, and passing through each deck, is a circular opening, designed as a shaft to carry the feed and water

pipes, and sufficiently large to permit the table operator to pass up and down and inspect the decks and feed boxes.

No formal tests have been made as yet, but preliminary sampling of the products indicates that the 20 decks are handling 140 tons (dry weight) of slime in 24 hours containing 2 per cent. copper, and producing a concentrate containing 7.25 per cent. copper, and a tailing containing 1.10 per cent. copper.   The recovery is about 53 per cent.
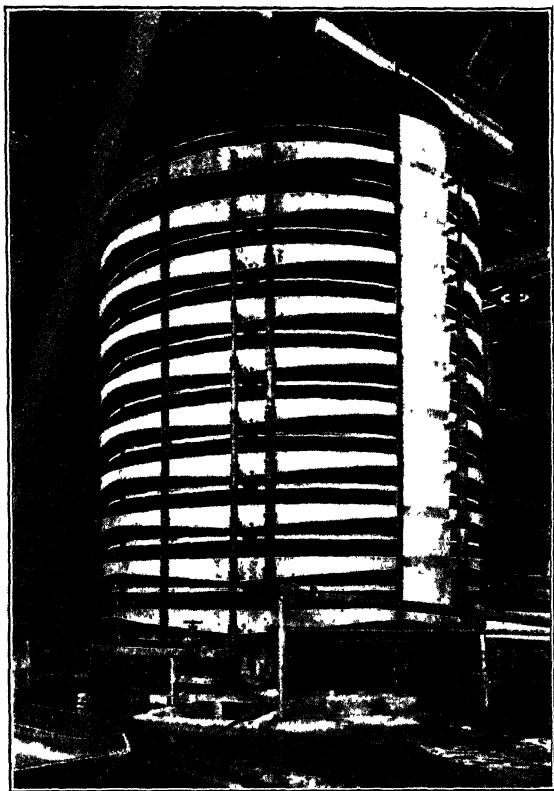
FIG. 9.—TWENTY-DECK ROUND TABLE.

Slime Feed Distributor.—In connection with test work in the slime plant, there has been devised a feeding device, Fig. 10, to give equal distribution of feed to any set of machines to which it is desired to send the same feed.   The same device is used as a distributor for the 20-deck round table.   It consists of two cylindrical tanks, one within the other, and the annular space between divided radially into as many equal parts as there are machines to be fed.   Each compartment is connected by a pipe to a machine.   Above the tanks

revolves a pipe with a 45° elbow on it, which distributes the feed to the several compartments. The top edge of the inner cylinder is slightly lower than the radial partitions, so that if one machine has to be closed down the feed which went to its compartment would overflow into the inner cylinder, which in turn would overflow all the other compartments, and the feed would still be equally divided. This device has been checked by time samples for tonnage, volume, and assay, and found to give very accurate distribution of the feed.
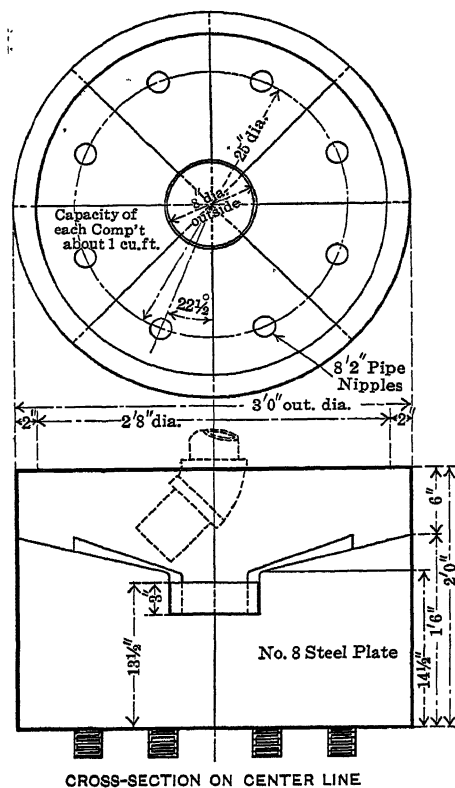
CROSS-SECTION ON CENTER LINE
FIG. 10.—FEED DISTRIBUTOR FOR SLIMES.

IX. CONCLUSION.

The data given in the preceding part of this paper are representative of the work which has been done on the problem. In many cases the figures given are averages of several tests or are tests of several days' duration. Moreover, operating tests have been cited where possible, as these are the conditions under which machines have to work in practice. There are therefore sufficient data to indicate the mode of procedure followed, and the results obtained in the attempt to solve the slime problem.

It is evident that large settling areas are required to thicken the slime where free settling is to be used. If filters or centrifugal machines are resorted to, then space can be economized, but increased maintenance costs and operation costs, together with complex machines, enter into the problem.

The Dorr tank has nearly the same capacity per square foot of settling area as the smaller units; namely, 0.20 to 0.25 gal. per square foot per minute; but it has the advantage of large units, say 50-ft. tanks, which can be built 3 ft. deep and three in a stand; and consequently requires few discharge plugs, and these of good size. The filter is a very compact and efficient machine, but has the disadvantage of operating in cycles as well as having high maintenance costs.

The Kuchs-Laist separator is very compact and is continuous in operation, but requires power, and also a satisfactory discharge plug to give constant density. Its product is freer of colloids than that of the other thickeners, which allows concentrators to make a greater net recovery from it. If a uniform spigot density could be maintained this machine would be very satisfactory.

The round table has proved very efficient for handling slime, it being simple in construction and operation, and a number of decks may be placed one above the other, thus economizing on floor space, where the mill height is available.

The Peck machine has a good principle, namely, the utilization of the effect of centrifugal force, which allows it to handle a good tonnage in a small space. It makes a good recovery on the slime settled in the ponds.

The best net recovery of copper in the original slime is from 50.0 to 52.0 per cent. except where the Kuchs-Laist machine is used. The slime contains about 20 per cent. of colloidal copper, which is not recoverable by wet concentration; and there are middling grains to be considered as well. Thus the net recovery of recoverable copper is probably 65 per cent., which is very good work for wet concentration on this class of material.

DISCUSSION.

JOHN V. N. DORR, Denver, Colo.:—Mr. Hayden has placed the metallurgical profession greatly in his debt for this valuable paper. The importance of such a complete description of the research work on which is based the building of a plant to add at least 10,000,000 lb. of copper per year to the production of the Anaconda Copper Mining Co., at small expense, will be appreciated by all our members, and especially those who are facing similar problems.

The information given in regard to the treatment methods discarded will be a great help to many workers, as it is often of as much importance to know what " not to do " as what method to adopt.

Some further comment on the tests of the Dorr thickener may be of interest, as they have led to developments which I believe will prove of great importance.

Before the Dorr thickener was installed, I had shown Mr. Laist reports from the Steptoe Valley concentrator, in which it was stated that one thickener 17 ft. in diameter was the equivalent in settling capacity of twelve 8-ft. cones, although the latter contained double the surface area. Mr. Laist had made ·careful tests on the settling capacity of an 8-ft. cone on Anaconda slime, and we naturally expected the thickener would show a much higher capacity for equivalent area.

When, as Mr. Hayden stated, the tests showed no increase, I realized that we were comparing it with a cone under test conditions, while the Steptoe results were based on operative conditions, and the explanation given by Mr. Waddell for the great difference had been, that " there is only one unit to watch instead of twelve, without the constant result of feed to twelve cones becoming unbalanced. Also the one 1½-in. spigot used on the Dorr requires no attention compared with the dozen ⅜-in. spigots of the cones."

I realized also as a result of the tests that the total settling area required to handle the immense tonnage of the Anaconda company would be enormous, and very expensive in view of the housing required by Montana climatic conditions. After studying the matter over, I told Mr. Laist that I believed a thickener operating in a much shallower tank would have nearly the same capacity. I asked him to test it out in the same tank filled 3 ft. deep, and stated that if my idea proved correct, we could manufacture thickeners to operate with one mechanism in a number of superimposed shallow tanks. As a result of the tests we are now manufacturing for the Anaconda company 40 of what we call four-deck thickeners, each to operate in four tanks 28 ft. in diameter by 3 ft. 3 in. deep. This will reduce to nearly one-fourth the size of the building required for housing the settlers, and save in operating costs as well. Based on power measurements at other plants, and assuming one man giving constant attention to the operation of the plant, the cost of clarification, omitting interest, should not exceed 0.1 c. per 1,000 gal. clarified.

Carrying the idea still further at that time, I devised a system of trays to be installed in deep thickeners, for the purpose of making in effect a plurality of shallow tanks, each with its own feed and with a

common overflow of clear liquid, and common or separate discharge of thick product.

A practical test along these lines at its mill has satisfied a large gold mining company, now cyaniding 1,800 tons of slime daily, that it can double its settling capacity with no additional expense for buildings, foundations, or tanks, and a very moderate cost for equipment. Both modifications of the thickener are now being patented, and I hope shortly to publish in the *Transactions* a full description of these recent developments.

The reduced cost of settling area thus obtained makes it probable that the machines can be used profitably in clarifying the water supply of cities depending on turbid river water, although I have no data as yet regarding the relative settling rates of a difficult metallurgical slime and river silt.

The installation at Anaconda should clarify enough water to supply a city of from 200,000 to 300,000 people, at the usual per capita rate of consumption.

Application of Hindered Settling to Hydraulic Classifiers.

BY EARL S. BARDWELL, GREAT FALLS, MONT.

In his paper entitled, Development of Hindered-Settling Apparatus, Dr. Richards has related the history of the development of the hindered-settling classifier and given illustrations of the several types of this apparatus which he has designed.[1] An earlier paper,[2] Close Sizing Before Jigging, shows the nature of the hindered-settling phenomenon and the results which it produces. In the first-mentioned paper Dr. Richards describes the method of getting hindered settling and shows the relation that the constriction bears to the classifier. The question of proper ratios between the area of the sorting column and the area of the constriction opening Dr. Richards leaves open, aside from certain statements as to the ratios which have been found to give good results in practice.

While working with a hindered-settling classifier of the type now in use at the Boston & Montana Reduction Plant of the Anaconda Copper Mining Co. a number of interesting facts were brought out which have thrown considerable light on the question of proper ratio between the area of the sorting column and the area of the constriction and which will be made the subject of this paper.

Free and Hindered Settling.—If a mineral particle be dropped into a vessel containing still water, the particle will sink in accordance with the laws of free settling in water, which are as follows:

1. Of two grains having the same specific gravity but differing in size, the larger will settle in still water at the greater velocity.

2. Of two grains having the same size but differing in specific gravity, the heavier grain will settle at the greater velocity.

From these laws it becomes evident that the velocity of settling depends primarily on the specific gravity and the size of the mineral grain. The fracture of the mineral may exert a very considerable

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[1] *Trans.*, xli., 421 to 439 (1910).

[2] *Trans.*, xxiv., 409 to 486 (1894).

influence on the settling velocity. Rittinger[3] has expressed the settling velocity of mineral grains by the formula, $V = K \sqrt{D\,(G-1)}$, where $V =$ velocity in millimeters per second, K is a constant dependent in part on the shape of the mineral grain as determined by its characteristic fracture, D is the diameter of the grain in millimeters, and G its specific gravity. Richards[4] has shown that this general formula holds true only within a specified range of sizes; in the case of quartz down to 0.2 mm.

If the mineral grain be subjected to the action of a rising current of water, as in a hydraulic classifier, we find that the settling velocity is exactly equal to the velocity of the rising current of water which will keep the grain poised, neither causing it to rise nor allowing it to sink.

If, instead of a single mineral grain, we drop a considerable number of grains into the vessel containing still water, we find that some of the grains are prevented from settling at their normal velocity by reason of the movement of neighboring grains. The grains are hindered in their efforts to settle, and this I shall term the hindered-settling effect. The same effect is observed to an even greater extent when the grains are subjected to the sorting action of a rising current of water, as in a hydraulic classifier. The action is similar to that which we might expect were the grains settling through a medium of higher specific gravity than that of water. The constriction which is placed in the sorting column of the hindered-settling classifier causes to be maintained in the sorting column a bed of grains, each grain of which is poised freely in the water. Dr. Richards has likened these grains to the grains of sand sometimes observed in similar motion in a boiling spring. This quicksand column acts as a filter, through which the other grains must make their way in order to be able to pass out of the apparatus through the spigot. The smaller grains of lighter material are unable to penetrate this bed to any depth and are consequently turned back, thus giving a spigot product with the high diametral ratios between average light mineral and average heavy mineral particles which are characteristic of hindered-settling classifiers.

Thus hindered settling may take place in a free-settling classifier when the classifier is overfed. The hindered-settling classifier differs from the free-settling classifier in that it is designed to make the best possible use of this hindered-settling effect and at the same time to furnish means of controlling and maintaining these conditions.

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[3] *Lehrbuch der Aufbereitungskunde*, p. 191 (1867).

[4] *Trans.*, xxxviii., 231 to 235 (1907).

The Constriction.—The constriction is the essential part of the hin-dered-settling classifier. Without some form of constriction at the base of the sorting column or teeter chamber no classifier can prop-erly be classed as a hindered-settling classifier.

The constriction may consist of a plate or diaphragm with a central circular opening of less diameter than that of the sorting column or with a series of smaller circular openings, or of a plate in which a series of short pipes or tubes have been inserted. The latter form of constriction has been found to give the best results when the classifier is designed to overflow quartz grains with a diameter of 0.75 mm. or larger. The former is sufficient in the case of deslimers.
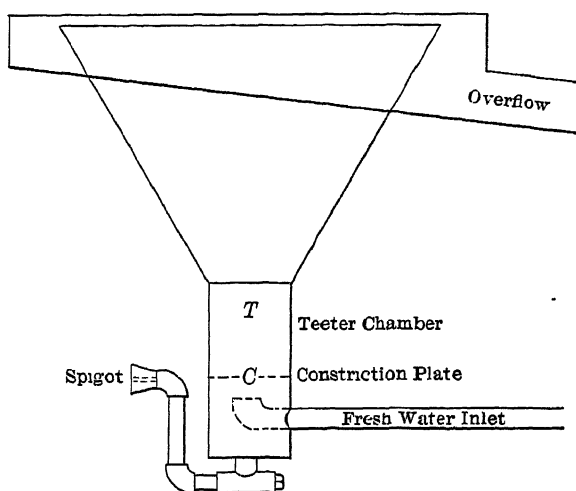
FIG. 1.—SINGLE-POCKET CLASSIFIER.

The effect produced by the constriction is as follows: It is evident from Fig. 1, which represents the type of ·classifier in use at the con-centrator of the Boston & Montana Reduction Works of the Ana-conda Copper Mining Co., that with a given volume of water rising through the constriction, C, the velocity through this constricted area will be greater than the velocity of the rising water in the teeter chamber, T; the velocities varying inversely as the respective cross-sectional areas.

Such being the case, grains of sand, the free-settling velocities of which are greater than the rising velocity in the teeter chamber, will sink and find their way into the teeter chamber, T. Such of these grains as have a free-settling velocity greater than the rising velocity through the constricted area will pass through the opening in the

constriction plate and out the spigot. Grains which are able to sink into the teeter chamber but are not able to sink through the swifter rising current in the constricted area accumulate in the teeter chamber and remain there poised freely in the water, moving among themselves and forming a quicksand bed which is in its effect somewhat analogous to a liquid having a higher specific gravity than water.

When grains have thus accumulated in the teeter chamber until a point is reached such that, in any cross-section taken through the sorting column or teeter chamber, the area occupied by sand is to the area occupied by water as the constricted area is to the area of the sorting column, then a condition of equilibrium is reached, which the classifier, if properly designed, will maintain. The classifier may now be said to have become bedded. The grains which now enter the classifier, and which are able, by reason of their higher settling velocity, to penetrate the teetering mass of grains in the teeter chamber, tend to upset this state of equilibrium and will either pass through the teeter chamber and out the spigot, or, if they have not a settling velocity sufficiently great, will, by increasing the density of the teetering column, cause a sufficient amount of material to be discharged through the spigot to restore the condition of equilibrium above mentioned. The bed or teetering mass of grains in an open-spigot classifier may be considered as composed of grains of a definite size; i. e., they are the grains which will sink into the teeter chamber and which are not able under free-settling conditions to pass through the opening in the constriction plate.

Observation of classifiers under operating conditions has shown that for each size grain there is a certain density that can be maintained in the teeter chamber, and that the ratio existing between the area of the sorting column and the area of the constriction opening depends primarily upon this permissible density. Dr. Richards points out that when the ratio is too large full teeter will take place in the sorting column, but no sand will be discharged through the spigot, while if the ratio is too small there will be no bed; in other words, the classifier will act by free settling.

With a closed-spigot classifier, such as Dr. Richards used in his experiments, this statement holds true. In the case of an open-spigot classifier it appears that if the ratio between the area of the sorting column and the area of the constriction opening be too large the result is a decrease in the capacity of the classifier and tendency to banking. There is a ratio which gives the maximum capacity obtainable from the classifier, and this ratio is the one arrived at by design-

ing the classifier to maintain a teetering column of maximum permissible density.

The permissible density which can be maintained with any given size of grains is that density which exists when all of the grains in the sorting column or teeter chamber are poised in the water, free to move up or down among themselves. This condition Dr. Richards has very appropriately called "full teeter." The density at full teeter varies, being greater when the bed of grains in the classifier is composed of large grains than when it is composed of finer particles. Experiments which will be described later seem to indicate that this permissible density varies inversely as the square root of the total surface exposed in a given weight of grains. This law apparently holds within the range of sizes that are usually handled by classifiers. There is an upper limit to the density that it is theoretically possible to maintain, and when this has been reached the theoretical ratio of the area of the sorting column to the area of the constriction opening becomes unity. Beyond this point we are unable to maintain a bed of teetering grains and the classifier acts by free settling. As this point would be reached in a classifier with which we wished to overflow 2.35-mm. material, it is of no practical consequence as far as the problem of hydraulic classification is concerned.

Permissible Density.—In order to determine what the full-teeter densities were the following experiment was tried. A quantity of tailings material was screen sized and the separate sizes treated in a sorting tube with closed spigot. This sorting tube was similar to the one used by Dr. Richards in his experiments. The weight of the grains of each size contained in a given volume at full teeter was determined. The densities which were obtained in this way formed a series varying from 4,618 g. to the gallon in the case of grains having an average diameter of 2.19 mm. to 866 g. to the gallon in the case of grains 0.07 mm. in size. A considerable number of relations were tried, but it was finally found that the relation that seemed to hold most nearly true was that the densities varied inversely as the square root of the total surface exposed in 1 g. of grains of each of the given sizes. The assumption was made that the particles were spheres with diameters equal to the average screen size. On this assumption the volume of one grain, the weight in grams of one grain, the number of grains in 1 g., the surface of one grain in square millimeters, and the total surface exposed in 1 g. of grains of each size were computed. Table I. shows the experimental results side by side with the computed results.

TABLE I.—*Permissible Densities of Quartz Grains in Quicksand Column.*

Diameter. Mm.	Volume 1 Grain. Cu mm.	Weight 1 Grain Grams.	Number Grains in 1 Gram	Surface 1 Grain. Sq. mm	Total Surface 1 Gram	Sq. Root Total Surface.	Density at Full Teeter. Grams per Gallon	Permissible Density.	Cu. mm. Sand in 1 Gallon.	Cu mm Water in 1 Gallon.	Ratio.
2.19	5.5	0.0146	69	15.05	1,040	32 2	4,618	4,830	1,830,000	1,955,000	1.07
1.84	3.26	0.00863	116	10.62	1,285	35.2	4,489	4,420	1,670,000	2,115,000	1.27
1 54	1.91	0.00506	197	7.45	1,466	38 4	4,239	4,050	1,520,000	2,265,000	1.49
1.20	0.903	0.00289	419	4.53	1,900	43.5	3,989	3,580	1,350,000	2,435,000	1.80
0 92	0 408	0 00108	926	2 65	2,460	49 6	3,747	3,140	1,190,000	2,595,000	2.18
0.841	0 311	0 000825	1,212	2.22	2,691	51 9	3,000	3,000	1,131,000	2,654,000	2.34
0.77	0.239	0 000635	1,580	1.86	2,940	54.2	3,353	2,870	1,085,000	2,700,000	2.49
0 60	0.113	0.000300	3,330	1.13	3,770	61.5	2,763	2,580	953,000	2,832,000	2.97
0 50	0 0655	0.000173	5,780	0.785	4,530	67.4	2,311	2,310	875,000	2,910,000	3.06
0.42	0.0388	0 000103	9,700	0.555	5,380	73.3	2,573	2,120	800,000	2,985,000	3.73
0.35	0.0224	0 0000595	16,800	0.385	6,470	80.4	1,937	1,940	729,000	3,056,000	4.20
0.26	0.0092	0.0000244	41,000	0.212	8,700	93.5	1,684	1,670	630,000	3,155,000	5 00
0.166	0.0024	0 00000635	15,7000	0 0866	13,600	117	1,333	1,380	504,000	3,281,000	6.51
0.070	0.00018	0.000000476	2,100,000	0 0154	32,300	180	866	870	327,000	3,458,000	10.57

In Table I. the line marked "Density at Full Teeter" gives the densities that were determined by experiment and are in fact the average of several results which agreed closely. The density in the case of grains 0.841 mm. in diameter being practically what we have found in practice, was assumed to be correct, and the values given in the line marked "Permissible Density" were computed on the assumption that the densities varied in the manner stated in the preceding paragraph. As will be seen, this relation seems to hold approximately true.

The ratio of area of sorting column to area of constriction opening may be computed from the formula $R = \dfrac{K}{\sqrt{D}} - 1$, where K is a constant approximately equal to 3.

We have so far considered the bed as composed of quartz grains of the size which would not ordinarily be able to penetrate the current rising through the constriction opening and which would not overflow under free-settling conditions. As a matter of fact, there must be some middlings grains and some heavy mineral grains in the classifier bed. This would not affect the results to any considerable extent. Wherever we have had occasion to use constrictions with these ratios in classifiers the classifiers have carried a bed and have given satisfactory spigot products. The curve, Fig. 2, covers the range of sizes with which we are ordinarily concerned in designing classifiers. The size or diameter in millimeters of the maximum quartz grain to be overflowed is plotted as abscissa and the corresponding ratio between the area of the sorting column and the

area of the constriction opening is plotted as ordinate. Thus in the case of a four-spigot classifier in which we wish to make the first spigot deliver 2 to 1.25 mm. quartz; the second spigot, 1.25 to 0.75 mm.; the third spigot, 0.75 to 0.35 mm., and the fourth spigot 0.35 to 0.10 mm. quartz, the ratio should be 1.8, 2.6, 4.2, and 8.9, respectively. To work properly, each pocket would have to be designed with reference to the tonnage of each of these sizes which the classifier might be expected to handle.

Having established the proper ratios for area of sorting column to area of constriction, it may be of interest to see how the other computations are made.

Size of Constriction Opening.—In designing a hindered-settling classifier the size of the constriction opening is the factor that determines the capacity of the classifier. It is, therefore, the part of the classifier which must first receive attention.
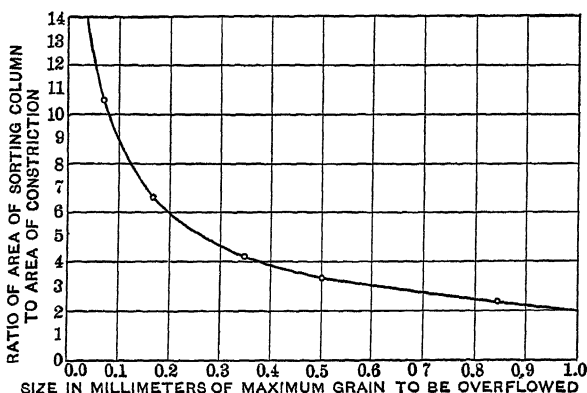
FIG. 2.—CURVE SHOWING PROPER RATIO OF CONSTRICTION TO BE USED IN SINGLE-POCKET CLASSIFIER IN ORDER TO OVERFLOW QUARTZ OF DIAMETER SHOWN BY ABSCISSA.

The minimum size of constriction opening depends upon the amount of spigot product that the classifier is to be called upon to deliver. If we know the average diameter of quartz grain which is to be in the spigot and the maximum tonnage of spigot product to be delivered, we would proceed as follows: From a table of settling velocities[5] find the average settling velocity of the average quartz grain above referred to. Now, from the tonnage calculate the volume that must pass the constriction in a given time; say, in one second. Divide the volume found in this way by the free-settling velocity of the average quartz grain, the former being in cubic inches and the latter in inches per second, and we have the area that would be re-

[5] *Trans.*, xxxviii., 229 (1907).

quired for the opening in order that it should just pass this volume of material. We have found from experience that this area must be multiplied by about 3, for reasons that will be stated later. This latter statement refers only to classifiers which have a rising current of water in the sorting column. The area found in this manner must now be multiplied by the proper ratio, as shown on the curve, Fig. 2, in order to obtain the area of sorting column that will be necessary.

Design of Classifier Top.—The top of the classifier should be so designed that the rising current at the top due to the combined effect of fresh water and feed water is equal or nearly equal to the rising current through the constriction or to the free-settling velocity of the largest quartz grain which the classifier is designed to overflow. If this is done, then the finer particles which we wish to keep out of the spigot, and which are not able to penetrate the bed of teetering grains in the sorting column, overflow at once. When the classifier is designed with a conical top or pocket and when the feed water quantity varies within wide limits it is practically impossible to so design a classifier as to enable it to do uniformly good work. When, however, the classifier is to treat the spigot of a deslimer and the volume of feed is kept constant the top can be so designed as to do very accurate sizing. In practice it has been found advantageous to feed a classifier of this type through a central feed cone so designed that the annular space at the bottom will give a rising current equal to the rising current through the constriction, with the annular space at the top approximately 25 per cent. greater than that at the bottom. In this way the full effect of the feed water may be utilized for the work of sorting.

Method of Designing a Classifier.—It may be of interest to outline the method that is used in designing a classifier of the general type illustrated in Fig. 1 so as to satisfy a given set of conditions. We will suppose a case in which we want a single-pocket classifier capable of desliming 500 tons per 24 hours, the material being the undersize of 5-mm. trommels. We will further assume that we have 8 ft. of head room, that we wish the deslimed product to have a density of 1,200 g. to the gallon, and that the feed to the classifier contains 10 per cent. of material finer than 200 mesh which we wish to overflow as slime. The feed water amounts to 300,000 gal. per 24 hours.

Size of Spigot.—In order that we may obtain a spigot discharge with a density of 1,200 g. to the gallon, the spigot discharge must amount to 340,200 gal. per 24 hours, $\dfrac{450 \times 2,000 \times 453.6}{1,200} = 340,200$. This

is equivalent to 3.938 gal. per second, or 910 cu. in. per second. With a 2.25-in. spigot, which has an opening equal to 3.97 sq. in., we would require a velocity of 910 ÷ 3.97, or 229 in. per second in order to obtain the above-mentioned spigot discharge. From the formula $V^2 = 2gh$, we can compute the head room theoretically required for the classifier. This comes out equal to about 68 in., or 5 ft. 8 in. This would be satisfactory, as we have allowed ourselves 8 ft. of head room. To allow for friction we would probably so construct the classifier as to have the overflow about 6 ft. higher than the spigot opening. This may be further modified when we come to design the classifier top.

Area of Constriction Opening.—If we are to make 450 tons per 24 hours of spigot product then we shall have:

$$\frac{450 \times 2,000}{86,400} = 10.42 \text{ lb. per second, or}$$

$$\frac{10.42 \times 1,728}{64.5 \times 3.1} = 90.05 \text{ cu. in. per second.}$$

This calculation is based on the assumption that the specific gravity of the material is 3.1.

We will assume that we have made a sizing test on the feed after first screening out the through-200-mesh material, and that we have obtained results as follows:

Screen Size.	Per Cent. Total.	Cumulative Per Cent.
On 5 mm.	0.6	0.6
2.38	23.2	23.8
1.41	23.5	47.3
0.841	19.2	66.4
0.500	13.3	79.8
0.350	4.2	84.0
0.166	10.6	94.6
0.078	5.4	100.0
Total	100.0	

The average screen size figures out 1.7 mm., and this grain has a settling velocity of 136 mm. per second, or, say, 5.35 in. per second. In order that 90.05 cu. in. of sand may pass the constriction with an average velocity of 5.35 in. per second the constriction opening must have an area of at least 16.8 sq. in. To allow for variations in feed we will make the constriction opening 5 in. in diameter, giving us an area of 19.6 sq. in.

Size of Sorting Column.—If we were making the calculations for a

classifier which was to be operated with a rising current and intended to overflow 200-mesh material we would multiply the constriction area which we have just found by 10.56, the ratio obtained from the curve, Fig. 2. This would necessitate a sorting column 16 in. in diameter. For a deslimer we have found that this does not seem to be necessary and that a 12-in. sorting column will answer the purpose. In the case of a deslimer the sorting column should be made at least 4 in. in length. The coarser the material which we wish to overflow, the longer should the sorting column be. A sorting column 8 in. long is, however, sufficient in most cases.

Area of Classifier Top.—A grain of quartz 0.078 mm. in diameter settles at the rate of 4 mm. per second. We will therefore make the rising velocity at the top of the classifier 3.5 mm. per second, in order to be sure that we overflow the least possible amount of on-200-mesh material. In a deslimer of this sort we need use only enough fresh water to supply the spigot, and all the water which will overflow is the water which comes to the deslimer with the feed. This we have stated amounts to 300,000 gal. per 24 hours, or $\dfrac{300,000}{86,400} = 3,472$ gal. per second, or 802 cu. in. per second.

If our rising velocity is to be 3.5 mm. per second, or 0.138 in. per second, we must have a top with an area of 5,812 sq. in. or a top with diameter of 86 in. If we make our cone 48 in. in vertical height, the classifier will have to be somewhat more than 6 ft. in vertical height, in order to allow for the sorting column, pressure chamber, and spigot discharge. About 7 ft. of head room would be required. As we have 8 ft. of head room, this will work out nicely. The classifier designed in this way will deliver its overflow with a density of 150 g. to the gallon.

As was stated previously, a classifier in which there is to be a rising current would be designed in a similar manner; the only difference being that when the size of the constriction opening has been computed the area found should be multiplied by about three. The reason for this is that with a given rising velocity through the constriction, and a definite volume of sand sinking through the constriction at a known average velocity, if the area of the constriction opening is made double the sand area the rising velocity is doubled and the classifier will bank or choke, since the rising velocity becomes greater than the average rate of settling of the grains in the spigot discharge. We will suppose a case in which the average settling velocity of the grains going into the spigot discharge is 170 mm. per second and where we have a rising velocity of 100 mm. per second.

In this case, if the sand area were doubled to get the constriction area the effective rising velocity would become 200 mm. per second. If the sand area be multiplied by three then the constriction area when sand is passing is reduced one-third and the rising current becomes 150 mm. per second, or may have this value momentarily. Now, the average settling velocity is 170 mm. per second, and experience has shown that under these conditions the classifier will operate satisfactorily and will handle its rated tonnage. It might in some cases be advisable to make the constriction opening a little larger in order to be on the safe side, but in order that we may make the most economical use of water the classifier should be made only sufficiently large for the work that it is called upon to do.

If the classifier is made too large for the amount of feed which it is to handle there will not be enough material coming into the classifier to maintain a bed and the classifier will tend to act by free settling. Classifiers designed in the manner outlined have been found to operate satisfactorily and to be extremely efficient as regards water consumption.

In the mills we have two variables which affect the operation of classifiers; namely, variations in tonnage and variations in density of feed. For this reason a classifier, to do its best work, should be especially designed to meet the conditions under which it is to operate. This is, however, impracticable from the point of view of the manufacturer of milling machinery, who is obliged to standardize the parts and confine himself to a few standard sizes. Of the two variables, the one which is the more serious as regards operation of the classifier is the variation in volume of feed. This can be made negligible in classifiers of this type by desliming the pulp previous to classification, the classifier receiving its feed directly as the spigot product of a' deslimer. In this way a few standard sizes may be designed so as to satisfy practically any condition which may arise in practice.

The Anaconda Classifier.*

BY ROBERT AMMON, GREAT FALLS, MONT

THE purpose of this paper is to present a brief sketch of the development of this hindered-settling classifier, but primarily to show the actual results obtained in practice with the classifier working on the Butte copper ores at the Boston & Montana Concentrator at Great Falls and the Washoe Concentrator at Anaconda. The writer has endeavored to be as brief as possible, giving only the essential figures showing the efficiency of the classifier and leaving considerable interesting information to be derived by the reader from the statistics given.

The paper deals with the practical work of the classifier under the following heads:

1. Desliming 4.0-mm. primary feed, overflow from 0.07 to 0.0 mm.
2. Desliming 1.25-mm. secondary feed, overflow from 0.07 to 0.0 mm.
3. Desliming Huntington Mill discharge, overflow from 0.07 to 0.0 mm.
4. Classifying 2.5-mm. deslimed secondary feed, overflow from 1.00 to 0.07 mm.
5. Classifying 2.5-mm. deslimed primary feed, overflow from 0.75 to 0.07 mm.
6. Classifying 4.0-mm. deslimed primary feed, overflow from 0.75 to 0.07 mm.
7. Classifying middling-tailing product from roughing tables, overflow coarse tailing.
8. Classifying 2.5-mm. trommel undersize, not deslimed, overflow from 0.75 to 0.0 mm.
9. Remarks on the inner feed cone.

In the following pages all overflows are given in quartz dimensions; i. e., by a 0.75-mm. overflow is meant an overflow whose maximum particle is a 0.75-mm. quartz grain. By the term "slime" is meant all material which will pass through a 200-mesh screen whose average opening is 0.07 mm.

The development of the classifier was commenced in the fall of 1910,

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* U. S. Patent No. 1,058,828, issued April, 1913, to A. E. Wiggin, A. E. Wheeler, and R. H. Richards.

and was primarily the result of a suggestion by Dr. Robert H. Richards, that a conical tank equipped with a pulsating current would deslime the feed to his direct pulsator classifier then installed in No. 4 Section of the Great Falls Concentrator. The success of this arrangement in removing the slimes from ore pulp, together with the demand for a deslimed feed to the Hancock jigs,[1] led to a series of experiments initiated by A. E. Wiggin in the fall of 1910. The final result of these and later experiments has been the production of the Anaconda hindered-settling classifier, which has as its chief assets the three leading factors to be reckoned with
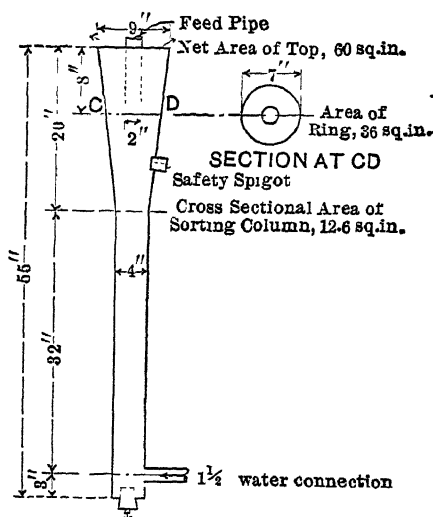
FIG. 1.—FIRST ANACONDA CLASSIFIER BUILT.

in any machine: namely, simplicity of construction, large tonnage, and high efficiency.

In criticizing the work of any classifier it must be remembered that we are dealing with grains of material of every conceivable size and shape from flat to round grains, and with varying specific gravities, from the purest and lightest gangue to the purest and heaviest mineral grains, together with the innumerable combinations of the same. If a classifier were mechanically perfect it still would be impossible to separate a large flat grain of gangue from a much smaller rounded grain of rich middlings.

Fig. 1 shows the first form of the classifier constructed. The early experiments proved that the pulsating current was of no advantage in this type of machine, hence it was abandoned for a plain rising current.

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[1] See paper to be presented at this meeting by Albert E. Wiggin, entitled, The Great Falls System of Concentration installed in Section No. 1 at the Washoe Concentrator, p. 209, this volume.

The classifier consisted of merely a vertical pipe for a sorting column with a superimposed inverted truncated cone, the hydraulic water being admitted at one side near the bottom of the sorting column at right angles to the spigot discharge.

The experiments immediately following consisted chiefly in varying the dimensions of the sorting column and the conical top, which will be referred to hereafter as simply the "top."

The object sought in these earliest forms was merely a deslimer which would give an overflow practically all through 200 mesh and at the same
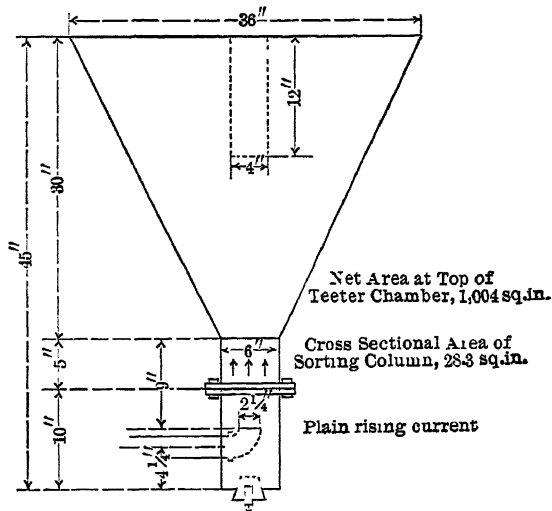
Fig. 2.—Anaconda Classifier. One of the Intermediate Forms.

time give a clear spigot free from slime. Many forms were tried. Fig. 2 shows one of the intermediate stages.

The hydraulic water was now admitted at 180° to the spigot discharge by means of a 90° elbow at the center of the sorting column. The work of this particular stage, while an improvement over earlier forms, was very crude when compared to the present form of deslimer.

Tons treated per 24 hr.................... 108
Per cent. total feed overflowed.......... . . .. 12.0
Per cent. solids in overflow on 200 mesh 45.7

LABORATORY TEST ON MODEL SIZE THREE-POCKET UNIT.

Fig. 3 is a sketch to scale of a unit of three pockets in series, built for the purpose of determining in the laboratory the relative velocities of rising currents required in the sorting columns to make the proper separations other than at 0.07 mm., and particularly at 1 mm., as this point

had been decided upon as the point of division between Wilfley table feed and Hancock jig feed in the mill flow sheet. The velocities shown on the sketch are the theoretical velocities and not the actual values. The actual velocities could not be determined in the experiment because of the lack of suitable means for measuring accurately such small quantities of water as were used by the laboratory classifiers. The feed to the unit consisted of the undersize of a 5-mm. round-hole trommel screen.
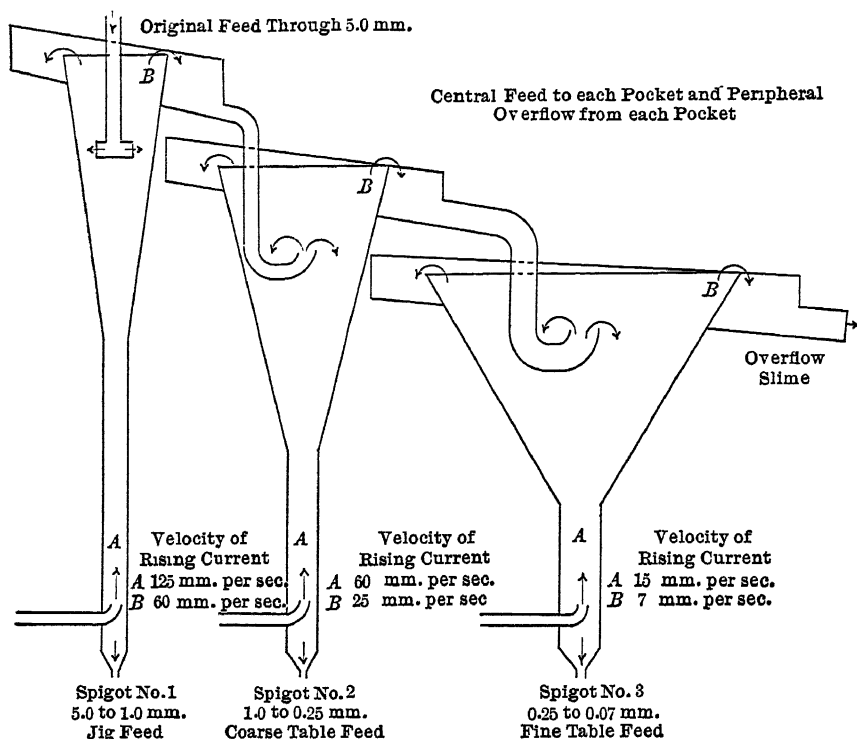
FIG. 3.—THREE-POCKET UNIT, LABORATORY SIZE.

The sand was fed dry to the first classifier by means of an automatic shaking feeder. The rising current in the sorting column of the first classifier was adjusted so that the dividing line between the first and second pockets was at about 1 mm. It was attempted to keep the overflow of the final pocket entirely through 200 mesh. The three spigot products and the overflow of the final pocket were screen sized, and each size was sorted into free mineral grains and quartz or middlings grains. The coarser sizes, from 4 mm. to 2 mm., were sorted by hand, while the finer sizes, between 2 mm. and 1 mm., were treated in a sorting tube, and the product below 1 mm. was concentrated on a laboratory size Wilfley table.

The amount of free mineral in the overflow (through 200 mesh) was estimated. The results of the test are shown in Table I and Fig. 4.

The classification chart, Fig. 4, which brings out the degree of classification quite distinctly, was worked out by C. R. Kuzell.

The free mineral overlaps on three screen sizes, 50, 60, and 90 mesh. The quartz grains in Spigot 1 which are finer than 1 mm. were fairly rich middlings grains whose specific gravities were greater than for those grains in Spigot 2 on 60 mesh, the latter being for the most part grains of quartz containing little or no mineral, and probably the bulk of these were flat grains. The feed was distributed as follows:

	Solids in Total Feed, Per Cent.	Free Mineral in Total Feed, Per Cent.
Spigot No. 1......	61.1	55.2
Spigot No. 2......	19 2	21 2
Spigot No. 3	3 3	4 1
Overflow......	16.4	19 5
TOTAL...	100.0	100 0

THE ADOPTION OF THE CONSTRICTION PLATE.

Up to this point no form of constriction had been used in the sorting column; and although a teetering mass had been observed in .the column at times, there is no doubt in the writer's mind but that this was due to overcrowded free-settling condition and not to true hindered settling. To produce strictly hindered-settling condition a constriction was adopted in the form of a thin plate of steel with a central circular opening. The result was to produce a teetering mass above the opening in the plate (sorting column), which will be called the teeter chamber. Later the depth of the sorting column was increased by using a nipple 2 in. long. The object of using a deeper sorting column was to do away with local eddy currents and to take up more or less the slight fluctuations in the hydraulic pressure. This lengthened sorting column increased the efficiency considerably, for at this point is determined whether a grain shall rise into the overflow or pass out the spigot. The teeter chamber had now been increased to 12 in. in diameter, and this diameter was adopted as a standard size.

From the theory of hindered settling[1] it is seen that the ratio between

[1] Development of Hindered-settling Apparatus, by R. H. Richards, *Trans.*, xli., 396 (1910).

Application of Hindered Settling to Hydraulic Classifiers, by E. S. Bardwell, p. 266, this volume.

TABLE I.—*Results of Test on Laboratory Three-pocket Classifier*

Screen Sizes (Millimeters) Through	On	Spigot No. 1 — Per Cent Solids in Spigot	Spigot No. 1 — Free Mineral, Per Cent Solids on Screen Sizes	Spigot No. 1 — Free Mineral, Per Cent Total in Spigot	Spigot No. 1 — Free Mineral, Per Cent Total in Feed	Spigot No. 1 — Per Cent Solids in Total Feed	Spigot No. 2 — Per Cent Solids in Spigot	Spigot No. 2 — Free Mineral, Per Cent Solids on Screen Sizes	Spigot No. 2 — Free Mineral, Per Cent Total in Spigot	Spigot No. 2 — Free Mineral, Per Cent Total in Feed	Spigot No. 2 — Per Cent Solids in Total Feed	Spigot No. 3 — Per Cent Solids in Spigot	Spigot No. 3 — Free Mineral, Per Cent Solids on Screen Sizes	Spigot No. 3 — Free Mineral, Per Cent Total in Spigot	Spigot No. 3 — Free Mineral, Per Cent Total in Feed	Spigot No. 3 — Per Cent Solids in Total Feed	Overflow — Per Cent Solids in Total Feed	Overflow — Free Mineral, Per Cent Solids in Total Feed	Overflow — Free Mineral, Per Cent Total in Feed
4.00	3.36	11.7	3.7	2.3	1.3	7.2													
3.36	2.83	10.1	4.3	2.3	1.3	6.2													
2.83	2.38	13.5	7.3	5.1	2.8	8.2													
2.38	2.00	8.6	7.6	3.4	1.9	5.3													
2.00	1.68	4.8	11.2	2.8	1.6	2.9													
1.68	1.41	12.2	14.7	9.4	5.2	7.4													
1.41	1.19	7.4	17.7	6.8	3.8	4.5													
1.19	1.00	6.3	21.6	7.1	3.9	3.8	4.3				0.8								
1.00	0.841	2.6	22.9	3.1	1.7	1.6	4.3				0.8								
0.841	0.707	7.4	26.5	10.2	5.6	4.5	4.0				0.8								
0.707	0.595	3.7	30.5	6.0	3.3	2.3	2.7				0.5								
0.595	0.500	1.7	32.2	2.8	1.6	1.0	6.4				1.2								
0.500	50 Mesh	2.8	51.0	7.4	4.0	1.7	7.1	7.3	2.2	0.5	1.4								
50 Mesh	60	2.1	65.0	7.4	4.1	1.3	13.4	15.4	8.9	1.9	2.6								
60 "	90	2.4	77.3	9.7	5.3	1.5	26.4	16.3	18.5	3.9	5.1	3.1				0.1			
90 "	120	2.7	100.0	14.2	7.8	1.7	9.0	30.8	11.8	2.5	1.7	2.0				0.1			
120 "	150						13.1	47.4	26.7	5.7	2.5	18.9	3.2	1.5	0.1	0.6			
150 "	200						3.8	72.7	11.9	2.5	0.7	11.8	5.2	1.5	0.1	0.4			
200 "							5.5	84.4	20.0	4.2	1.1	64.2	39.7	97.0	3.9	2.1			
Total		100.0		100.0	55.2	61.1	100.0		100.0	21.2	19.2	100.0		100.0	4.1	3.3	16.4	25.0	19.5

The entire chart area represents the total feed to the classifiers.

The area covered by each conventional sign represents the proportion of the original feed distributed to each pocket and to the overflow.

The closely hatched area represents the amount of free sulphide mineral.

FIG. 4.—TEST ON A SERIES OF THREE SINGLE-POCKET CLASSIFIERS (LABORATORY SIZE). CLASSIFICATION CHART BASED ON SCREEN SIZING AND SORTING TESTS.

FIG. 5.—STANDARD 7-FT. DESLIMING CLASSIFIER. CROSS-SECTION.

the cross-sectional areas of the teeter chamber and the sorting column is of the utmost importance. If the ratio is too small, free-settling classification is the result; if too large, the classifier will bank or choke up; while intermediate between these two ratios is one which will establish equilibrium between the teetering mass and the discharge through the sorting column. With this ratio established for a given condition, the classifier will maintain a bed of quicksand above the constriction, with
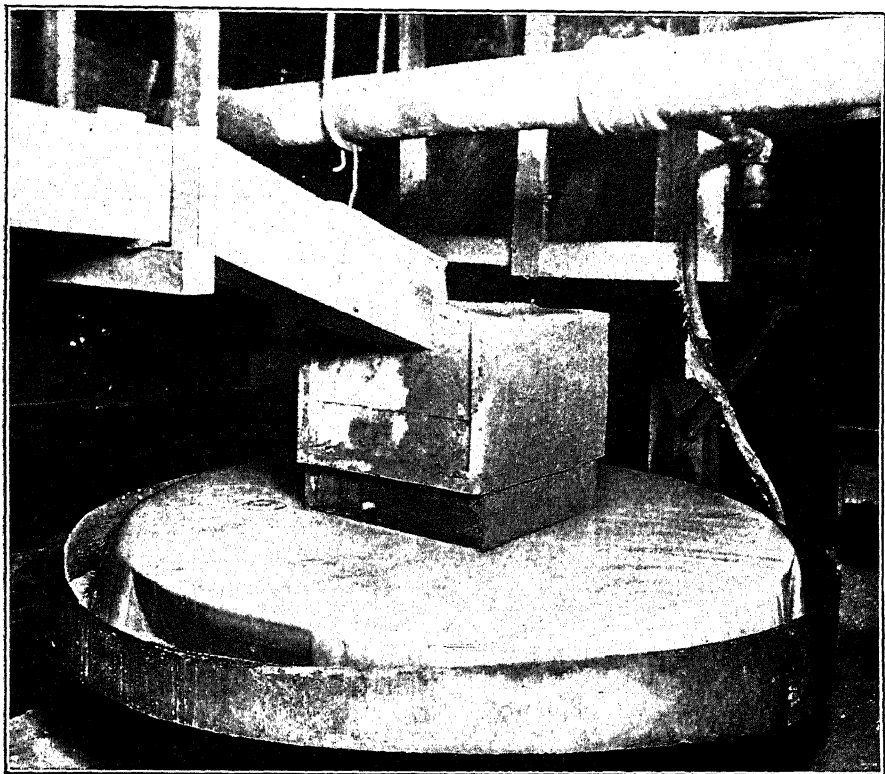
FIG. 6.—DESLIMER IN USE AT B. & M. CONCENTRATOR.

a constant discharge through the spigot. Numerous interesting experiments have been followed out along this line, with the result that the ratios given in the following pages have been found best. The single opening has been discarded for numerous smaller openings, with the idea of equalizing the rising current throughout any given cross-section of the teeter chamber.

The present form of the classifier is installed in six of the eight sections of the Great Falls Concentrator and the remodeled Section No. 1 of the Anaconda Concentrator, of which it may be said to form the nucleus.

The Classifier in Actual Practice.

The classifiers may be divided into two groups, according to the work performed: namely, desliming classifiers and table-feed classifiers.

Desliming Classifiers.

Fig. 5 gives a cross-sectional view of a standard 7-ft. deslimer, showing the essential point of construction.

Fig. 6 is a photographic view of a deslimer in use at the B. & M. Concentrator, Great Falls. These deslimers are used for desliming pulp from 4 mm. down and have proved very efficient. One particular feature which has been found to hold in practice is that when desliming fine pulp (2.5 mm. or under) no rising hydraulic water is used in the classifier; i. e., the only fresh water used in the deslimer is that used to supply the spigot, which is usually about 50,000 to 90,000 gal. per 24 hr. However, in desliming coarser feed some rising water is generally used. The B. & M. practice is to use a constriction of one 6 in. diameter by 2 in. long pipe, giving a ratio of 3.9. The Anaconda practice is to use a number of smaller pipes with a ratio of 4.6. Which is the better cannot be said, for conditions of practice are considerably different at the two mills. The shortest and best way to show what the classifiers will do in practice is to quote actual figures, obtained under different conditions.

If fed under ideal conditions a 7-ft. deslimer will handle 250,000 gal. of pulp with 200 tons of solids, giving an overflow of not over 2 per cent. of total solids in the overflow on 200 mesh. The spigot will be perfectly clear and very dense.

1. Desliming 4.0 mm. Primary Feed, Overflow from 0.07 to 0.0 mm.

"Primary" feed contains all the original mine fines and no finishing-roll product. Of necessity it is a very rich feed and contains a large amount of free mineral.

Following are figures representative of the work done at Great Falls on the above class of feed. The sum of the plug and the overflow is taken as the feed, which in all cases checked the actual feed sample.

	Gallons per 24 hr.	Pounds per 24 hr.	Per Cent. Total Feed	Density, Per Cent. Solids	Assay, Per Cent. Copper
Spigot	110,416	320,972	80.4	28.3	2 90
Overflow	205,200	78,160	19.6	4.4	2.54
Feed	134,616	399,132	100.0	28.7	2 83
Total, fresh water	181,000	....	....	... .	....
Rising water	83,424	....	....	....	....

Gallons of water consumed per ton of feed, 905.
Diameter of spigot discharge, 1.25 in. Spigot clear.
Rising velocity through constriction = 226 mm. per second.
Ratio of cross-sectional area of teeter chamber to sorting column = 3.9.
Constriction opening = one pipe 6 in. in diameter, 2 in. long.

Screen Sizing Figures.

SCREEN SIZES, Millimeters		SPIGOT		OVERFLOW	
		Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
Thru	*On*				
..	4 00	1 9	1 9	... .	... .
4 00	2.38..	22 0	23 9	....	. ...
2 38	1.68..	16 9	40 8	....	....
1.68	1.41...	6 9	47 7	....	....
1.41	0.841......	18 9	66 6	....	....
0.841	0.500....	12 6	79 2	. ...	. .. ...
0 500	0.350....	4 5	83.7	... .	... .
0.350	0.166............... . ..	9.7	93 4	....	... .
0.166	0.07....	5 6	99.0	7.9	7.9
0.07	....	1.0	100.0	92.1	100.0
	TOTAL........	100.0	100.0	100.0	100.0

The overflow is shown as containing 7.9 per cent. of total overflow on 200 mesh, which is no doubt due to excessive use of hydraulic water. Under test condition this same deslimer has shown 2.3 per cent. on 200 mesh, which is normal. However, as these and all figures following, except where otherwise stated, are actual practice figures and not test figures, the writer will explain from his personal observations and experience with the classifier such abnormal figures as might be misleading to the reader unfamiliar with the classifier.

The same deslimer when treating a little coarser feed gave as follows:

	Gallons per 24 Hours	Pounds per 24 Hours	Per Cent. Total Feed	Density, Per Cent. Solids
Spigot.....	111,104	222,221	91.4	20.7
Overflow....	196,340	21,600	8.6	1.3
Feed...	205,229	243,821	100 0	15.5
Total, fresh water	102,215	....	....	....
Rising water	None	....	....	....

Gallons water consumed per ton of feed, 840.
Diameter of spigot, 1.25 in. Spigot, clear.

The overflow screen sized 6.6 per cent. of total solids on 200 mesh. The water consumption is low, but could have been made considerably lower by running a denser spigot.

2. Desliming 1.25 mm. Secondary Feed, Overflow from 0.07 to 0.0 mm.

In this case the "secondary" feed consisted of the undersize of 1.25 by 12 mm. punched-plate trommels screening finishing-roll product. The feed was very dense and afforded excellent feed for a deslimer.

	Gallons per 24 Hours	Pounds per 24 Hours	Per Cent. Total Feed	Density, Per Cent. Solids
Spigot . .	52,010	272,160	96 0	44.2
Overflow	85,790	11,230	4 0	1.6
Feed..	79,800	283,390	100 0	33.1
Total, fresh water. . .	58,000	. .	. . . . .	. ., ...
Rising water . . .	16,878	. . . .	. . . . . .	. . . . . .

Gallons water consumed per ton of feed, 409.
Rising velocity through constriction, 84 mm. per second.
Diameter spigot 0.75 in. Spigot contains trace of slime.

Screen Sizing Figures.

SCREEN SIZES, Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 1 41. ..	8 4	8 4	8.4	8.4	. . . .	. . . .
On 0 841 . .	38 1	46 5	41.1	49.5	. . . . . .	. . . . . .
On 0 500 ...	29.0	75.5	29.1	78.6	. . . . ..	. . . . . .
On 0 350.. ..	5.4	80 9	6.1	84.7	. . . .	. . . . . .
On 0 166 ...	9.5	90.4	9.0	93.7	. . . .	. . . .
On 0.07. ..	4.5	94.9	4 3	98.0	0 2	0.2
Through 0 07.	5.1	100 0	2 0	100 0	99 8	100 0
TOTAL....	100.0	100 0	100 0	100 0	100 0	100 0

The above results were obtained from one of the secondary deslimers in use at Anaconda running under normal conditions. The constriction opening in this case was 11 1.5-in. nipples 3 in. long giving a ratio between the area of the teeter chamber and the sorting column of 4.6.

3. Desliming Huntington Mill Discharge, Overflow from 0.07 to 0.0 mm.

The feed was the discharge of one or more 5-ft. Huntington mills clothed with 1 by 12 mm. punched-plate screens. The practice was to deslime the above feed and treat the spigot on finishing Wilfleys. The writer has observed tables fed in this manner handle close to 50 tons per 24 hr.,

and make 80 per cent. of the feed into 0.45 per cent. Cu tailings. One of the reasons for this splendid work is the distinct valley produced on the table as the hindered-settling effect of the classifier, resulting in the clean separation of free mineral and gangue grains. Oftentimes this valley was from 10 to 12 in. wide and extended the entire length of the diagonal division line between the mineral and the gangue. Very little middlings was made on these tables. I shall give two sets of figures under this type, showing a medium volume of feed and an excessive volume of feed.

	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density Per Cent. Solids	Assay Per Cent. Copper
Spigot.	118,300	307,108	75.0	25 8	1.01
Overflow...	539,000	102,200	25 0	2.3	1 37
Feed	536,100	409,308	100.0	8 6	1.10
Total fresh water	121,200	... .	... .	....	....
Rising water..	15,184	....	.. ..	....	. ..

Gallons water consumed per ton of feed, 590.
Diameter of spigot, $1\frac{1}{8}$ in. Spigot, clear.
Constriction opening, one 6-in. pipe 2 in. long.
Constriction ratio, 3.9.
Rising velocity through constriction = 64 mm. per second.

Screen Sizing Figures.

SCREEN SIZES Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 1.41	3 4	3.4	5.4	5.4	....	. ...
On 1.00	9.0	12.4	11 7	17 1	....	......
On 0 841	13.5	25.9	16.0	33 1	....	.... .
On 0 707	10 9	36 8	13.9	47 0	....	.... .
On 0 500	11 7	48.5	13.2	60 2	..... .	....
On 0 350	9 9	58.4	14.7	74 9	....	....
On 0 166	12.4	70.8	15.5	90.4	....	....
On 0 07	8 7	79.5	8.6	99 0	9.4	9.4
Through 0 07	20.5	100.0	1.0	100.0	90.6	100 0
Total.....	100 0	100 0	100 0	100.0	100 0	100.0

The excessive fine material in the feed, 20.5 per cent. through 200 mesh, is due to the use of roll-product slime as sluicing water in the mill aprons.

Following are figures showing the work of the same deslimer under a much smaller volume of feed pulp:

	Gallons per 24 Hr	Pounds per 24 Hr.	Per Cent. Total Feed	Density Per Cent. Solids
Spigot..	106,596	177,120	80 1	15 0
Overflow.	186,120	44,150	19.9	2 7
Feed	182,716	221,270	100.0	11.6
Total fresh water..	110,000	. ..	... . . .	....
Rising water..	10,489	....	.. . . .	....

Gallons fresh water consumed per ton of feed, 1,000.
Same size spigot and constriction as above.
Rising velocity through constriction = 41 mm. per second.

Screen Sizing Figures.

SCREEN SIZES Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 1.41	5 9	5 9	5 1	5 1	....	....
On 1 00	12.1	18.0	12 0	17 1	....	... .
On 0.841	15.9	33.9	18 5	35 6	. . ..	....
On 0 707	9.8	43.7	13 0	48 6	. ...	....
On 0 500	9.7	53 4	13 5	62 1	... .	....
On 0.350	6.2	59.6	7 9	70 0	.. ..	.. . ...
On 0.166	10.9	70 5	14.7	84.7	.. ..	....
On 0.07	6.7	77 2	10 9	95 6	1.2	1.2
Through 0 07	22.8	100.0	4.4	100.0	98.8	100.0
Total.....	100.0	100 0	100 0	100.0	100.0	100.0

In desliming classifiers the volume of the feed pulp is the most important factor. Experiments have shown that with a deslimer with an 84-in. diameter top the pulp volume should never exceed 250,000 gal. to make an overflow 100 per cent. through 200 mesh. The reason for this is apparent.

Effective cross-section near overflow level = 5,290 sq. in.
Overflow = feed volume plus rising water.
Rising water = none.
Overflow = 250,000 gal. per 24 hr.

$$\frac{250,000 \times 231 \times 25.4}{86,400 \times 5,290} = 3.2 \text{ mm. per second rising current near overflow level.}$$
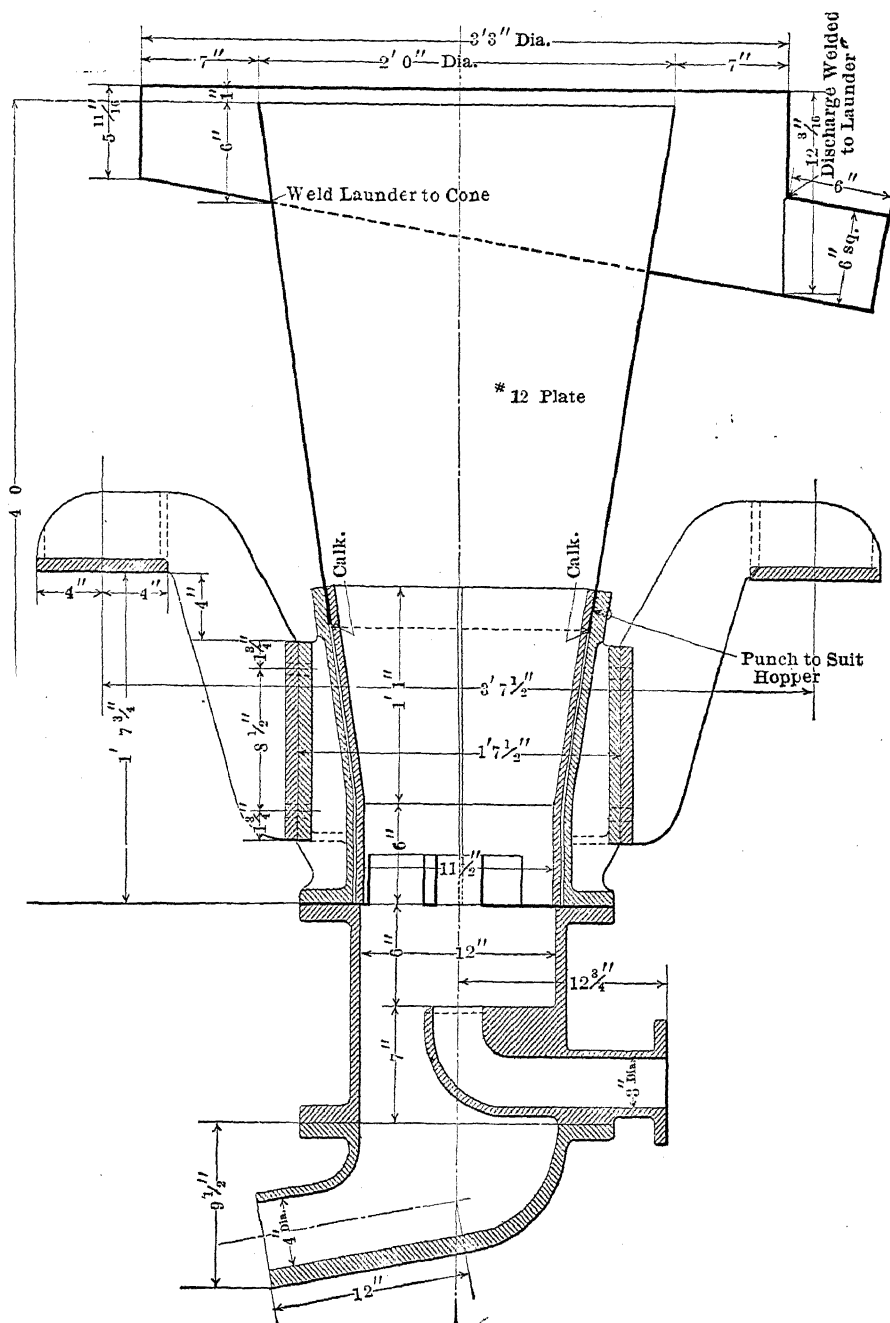
FIG. 7.—TABLE-FEED CLASSIFIER AT WASHOE CONCENTRATOR. CROSS-SECTION.

Richards[1] gives the free-settling velocity of quartz whose average diameter is 0.0747 mm. as 3.57 mm. per second. The feed must be constant, however, if we expect to use no rising water in the classifier.
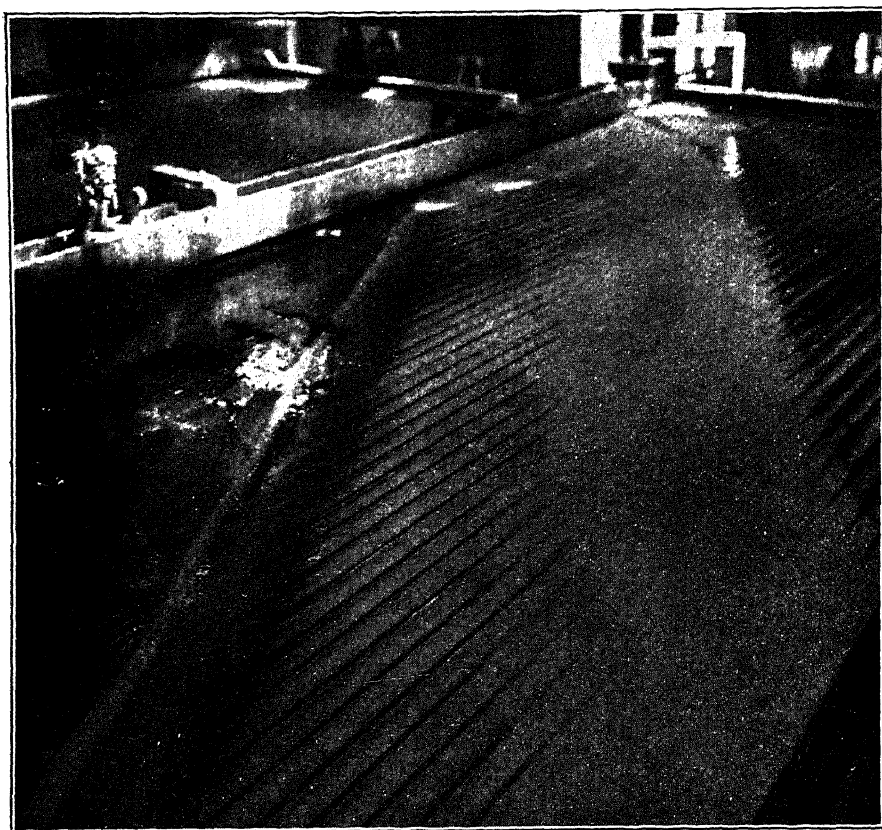

FIG. 8.—VIEW OF TABLE-FEED CLASSIFIER.

From the foregoing, the following conditions are most favorable for desliming 2.5 mm. pulp:

1. Maximum feed volume, 250,000 gal.
2. Maximum feed density, 17 per cent. solids.
3. Maximum tonnage, 250; average tonnage, 200.
4. Rising velocity at overflow level, 3.2 mm.
5. Rising velocity through constriction varies with the size of material, from close to zero to 40 mm.

For material coarser than 2.5 mm. the density of pulp through the sorting column (constriction) must be lower, to prevent the possibility of choke-ups. As a consequence a rising current must be maintained in

[1] *Ore Dressing*, vol. iii, p. 1422 (1909).

the sorting column or slime will enter the spigot. The finer pulp (2.5 mm. or below) seems to act as a filter for the slime feed and oftentimes results have been reported showing the extraction of deslimed water from the
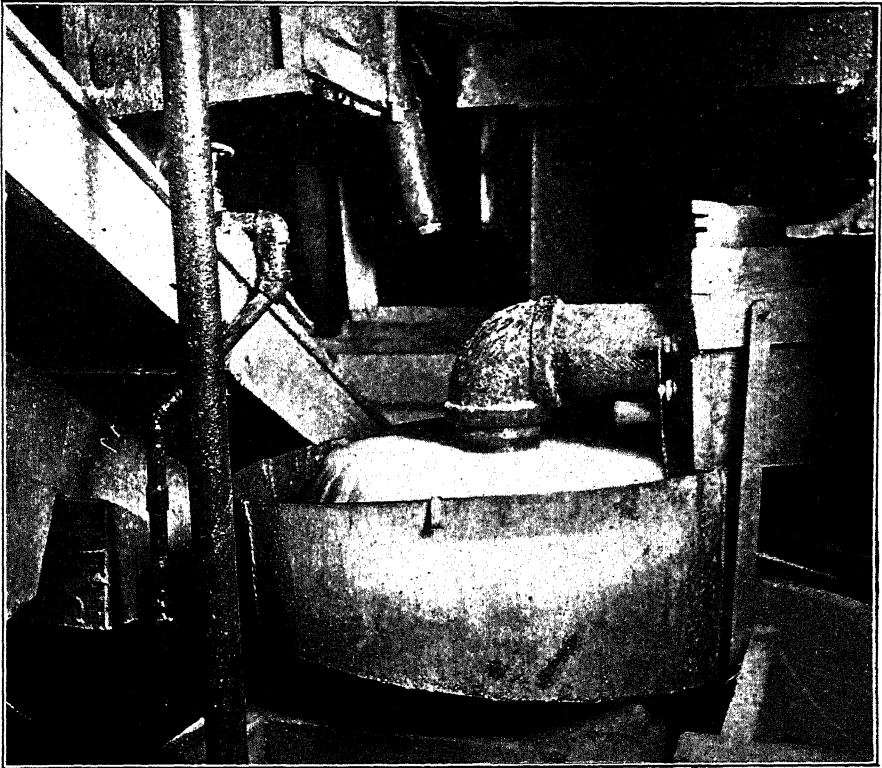

FIG. 9.—OVERFLOW OF TABLE-FEED CLASSIFIER.

feed and the addition of the same to the spigot. This can only be accounted for by the filtering effect of the teeter chamber. With feed above 2.5 mm. this phenomenon has seldom been observed.

TABLE-FEED CLASSIFIERS.

The second division of these classifiers are called table-feed classifiers because their function is to overflow a product which may be best concentrated on some type of shaking table. The chief use to which this classifier is put is to overflow a 0.75 mm. table feed, either primary, or secondary, or mixed.

Fig. 7 is a cross-section of one of the table-feed classifiers, of the type installed at the Washoe Concentrator. Unfortunately, no figures could be obtained showing the work of these classifiers in practice. At the

B. & M. Concentrator an inner cone has been added to the original design and has found universal use in this type of classifier. The development of the feed cone was begun in May, 1912, the first classifier with an inner
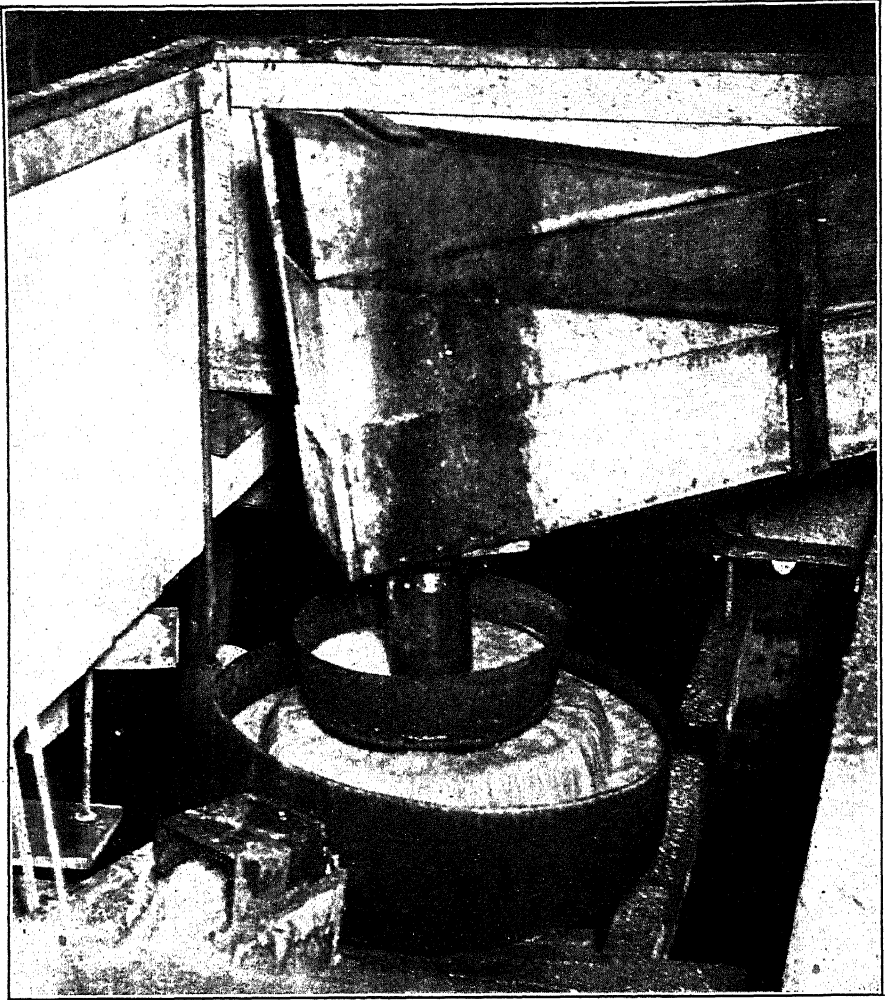

FIG. 10.—CLASSIFIER WITH INNER FEED CONE.

cone being installed, I believe, on May 9, 1912. Although we have no figures showing the classification of the Anaconda classifiers at the Washoe Mill, Fig. 8 gives a very clear conception of the actual work done by them. Dr. Richards states in his paper, The Development of Hindered-Settling Apparatus, that the advantage of hindered-settled feed on a shaking table is the production of a valley between the mineral and the sand, thereby aiding the separation of the different products.

FIG. 11.—SPIGOT OF TABLE-FEED CLASSIFIER.

In Fig. 8 we see this valley as obtained in actual practice. The photograph was taken after shutting off the feed and dressing water simultaneously and then shutting down the table approximately 10 sec. after the feed and water were cut off. The table was handling from 15 to 18 tons of feed per 24 hr. The tailings at the deepest point were 0.75 in. deep, the concentrates 0.25 in. deep, and the middlings about $\frac{1}{16}$ to $\frac{1}{8}$ in. deep. The tailings will probably assay 0.30 per cent. of copper. The feed to the table ranged from 0.75 to 0.07 mm. and was overflowed from the classifier shown in Fig. 8. The apparent broken spot on the Wilfley is due to reflections.

4. Classifying 2.5 mm. Deslimed Secondary Feed, Overflow from 1.00 to 0.07 mm.

The feed was deslimed rolled product through 2.5-mm. round-hole trommel. The overflow was heavy and classification not very close   The classifier was one of the type shown in Fig. 9, no inner cone being used.

	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density Per Cent. Solids
Spigot	129,254	203,472	37.7	16 7
Overflow........	408,205	335,750	62.3	9 2
Feed	223,434	539,222	100.0	24.2
Total fresh water.....	314,025	....	. .....	....
Rising water.......	192,906	. ...	......	.

Gallons water consumed per ton of feed, 1,165.

Screen Sizing Figures.

SCREEN SIZE Millimeters	SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 2.38.....................	2.6	2.6	....	....
On 1 41.....................	34.3	36.9	3.4	3.4
On 1.00.....................	27.7	64.6	8.4	11.8
On 0.841....................	21.2	85.8	17.4	29.2
On 0.500.......	12.0	97.8	33.5	62.7
On 0.350....................	2.2	100.0	10.0	72.7
On 0.166..	....	.. ..	18 9	91.6
On 0 07.......	....	....	6 1	97 7
Through 0.07.................. ..	... ..	....	2.3	100 0
Total..	100.0	100.0	100.0	100.0

Fig. 9 shows the overflow of the above classifier.

5. Classifying 2.5 mm. Feed, Overflow 0.75 mm.

The classifier was equipped with an inner feed cone (Fig. 10).   The feed consisted of 2.5-mm. round-hole trommel undersize, deslimed.

	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density Per Cent. Solids	Assay Per Cent. Copper
Spigot ..	116,000	314,064	62 8	21.5	2 56
Overflow ..	338,408	186,278	37 2	5.9	2 88
Feed...	175,300	500,342	100 0	21.8	2 69
Total fresh water .	279,108	. ..	.	... ..	. ..
Rising water... . . .	175,671	. .	.	.....	....

Gallons water consumed per ton of feed, 1,100.
Diameter of spigot, 1.25 in.
Constriction opening, 12 2-in. pipes 2 in. long, or 40.0 sq. in.
Ratio of area of teeter chamber to sorting column, 2.81.
Rising velocity through constriction = 316 mm. per second.
Rising velocity at bottom of annular space = 144 mm. per second.
Rising velocity at top of annular space = 122 mm. per second.

Screen Sizing Figures.

SCREEN SIZES, Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 2 00	7.4	7.4	12 3	12 3	....	.. ..
On 1 68	12.0	19.4	18 0	30.3	....	....
On 1.41	11.1	30 5	13 8	44.1	.. ..	. .....
On 1 00	13.4	43.9	21.0	65.1	.. ..	....
On 0.841	11.7	55.6	14 8	79.9	6.7	6.7
On 0 707	7.1	62 7	9 2	89.1	6.1	12.8
On 0.500	7.6	70.3	6 1	95.2	11.0	23.8
On 0 350	7.5	77.8	2 9	98.1	16.0	39.8
On 0.166	10.5	89.3	1 9	100.0	27.8	67.6
On 0 07	8 9	98 2	....	....	26.1	93.7
Through 0.07	2 8	100.0	....	....	6.3	100.0
Total.....	100.0	100.0	100 0	100.0 .	100.0	100.0

The figures show this classification to be excellent, although the classifier was designed to receive a feed of 250,000 gal. per 24 hr., whereas it is receiving only 175,000 gal.   In the spigot fully 80 per cent. of the fine material, through 0.707 mm., is free mineral grains, 'which we would expect in any spigot, even the perfectly classified.

A second set of figures shows the work performed on a feed containing more fines.

	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density, Per Cent. Solids	Assay, Per Cent. Copper
Spigot	115,300	283,392	47 8	19 3	3 15
Overflow...............	356,174	310,348	52 2	8 9	4 58
Feed...	158,300	593,740	100 0	25 7	3 89
Total fresh water.......	313,174	.. . .	. . ....	. . ...	. ...
Rising water	209,210		... .	....	.. . .

Gallons water per ton of feed, 1,050.
Spigot, 1.25 in. in diameter.
Constriction area, 40 sq. in.
Constriction ratio, 2 81.
Rising velocity through constriction = 373 mm. per second.
Rising velocity at bottom of annular space = 152 mm. per second.
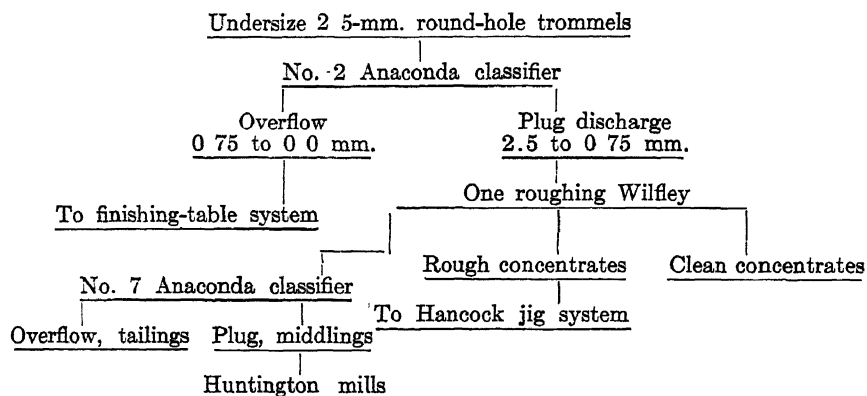Rising velocity at top of annular space = 129 mm. per second.

Screen Sizing Figures.

SCREEN SIZES, Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 2 00	7 2	7 2	13.9	13 9	.... .	. . . .
On 1 68	11 8	19 0	23 6	37.5	... .	. . ..
On 1.41	7 2	26 2	16.5	54 0	.. .	. .....
On 1.00	12 9	39.1	20.3	74 3	.. .	.....
On 0.841	9 6	48 7	13.6	87.9	7 6	7.6
On 0.707	8 8	57.5	6.9	94 8	5 6	13.2
On 0.500	8 3	65.8	3.8	98.6	8.7	21 9
On 0.350	10 6	76.4	1 4	100 0	16.6	38.5
On 0.166	12 0	88 4	... .	. ... .	29 9	68 4
On 0.07	9.0	97.4	.... .	.	26 7	95.1
Through 0.07	2.6	100 0	..	. . ..	4.9	100 0
Total.....	100.0	100.0	100 0	100 0	100 0	100.0

The work done by this classifier, as shown by the two sets of figures, may be said to be the best obtained with any type of the Anaconda classifier. The separation at 0.75 mm. is remarkable and shows the possibilities of the machine. This particular unit is used at the B. & M. Concentrator to furnish Hancock jig feed and finishing-table feed. Figs. 10 and 11 show the overflow and spigot of this installation as in use to-day. Note the appearance of the overflow in the photograph. It is very even except for the agitation produced by the inner cone throwing backwards into the overflow, material which is carried up the outside of the cone.

6. Classifying 4.0 mm. Original Feed, Overflow 0.75 mm.

The feed was deslimed original material, being the undersize of 5.0 mm. round-hole trommels. Classifier equipped with feed cone.

	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density, Per Cent. Solids	Assay, Per Cent. Copper
Spigot	97,292	420,076	65.0	27 9	3.24
Overflow.	253,806	224,994	35 0	9.1	3 42
Feed...	207,708	645,070	100 0	22.9	3 30
Gallons fresh water .	143,390	....	.... . .	.... ..	.... . .
Rising water..........	62,898	....	. ....	.. ...	. . ...

Gallons water per ton of feed, 445.
Spigot diameter, 1.25 in.
Constriction area, 40 sq. in.
Constriction ratio, 2.81.
Rising velocity through constriction = 136 mm. per second.
Rising velocity at bottom annular space = 109 mm. per second.
Rising velocity at top annular space = 92 mm. per second.

Screen Sizing Figures.

SCREEN SIZES, Millimeters	FEED		SPIGOT		OVERFLOW	
	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids	Individual Per Cent. Solids	Cumulative Per Cent. Solids
On 4 00	1.9	1.9	2.7	2.7	....	....
On 2.38	22.0	23.9	28 0	30.7	.. ..	....
On 1.41	23 8	47.7	30.2	60.9	... ...	....
On 0.841	18 9	66.6	22.8	83.7	7 3	7.3
On 0.500	12.6	79.2	7.8	91 5	16 4	23 7
On 0.350	4.5	83.7	4.2	95 7	8 9	32 6
On 0.166	9 7	93.4	3.7	99 4	33.0	65.6
On 0.07	5.6	99.0	0.6	100 0	25 8	91 4
Through 0.07	1.0	100 0	....	....	8 6	100.0
Total.....	100.0	100 0	100.0	100 0	100.0	100.0

7. Classifying Middlings-Tailings Roughing-Table Product, Overflow Coarse Tailings.

At the present time there are no classifiers in the B. & M. Concentrator overflowing tailings. This is due chiefly to the lack of head room, as the mill is old and not designed along modern lines.

The B. & M. system of concentration as installed at the Washoe mill includes four of these classifiers in a 1,500-ton unit, although two of

them only are used most of the time. Fig. 12 is a view of one of the tailings classifiers in use at the Washoe mill. The classifier in this case is discharged through the lower spigot direct to Huntington mills. The feed is the middlings-tailings product from the roughing tables and the overflow averages 0.45 per cent. of copper. Two spigots are provided for the classifier, the original idea being to use one in case the other choked. However, it has been found unnecessary to provide the second spigot, the later types being cast with but one (see Fig. 7). The following data are not taken from regular practice but are taken from a several days' test run and suffice to show what the classifier will do under this class of feed. The so-called middlings-tailings consist of true middlings grains and coarse tailings grains with no free mineral. The test was carried out according to the following flow sheet:

Undersize 2 5-mm. round-hole trommels

No. 2 Anaconda classifier

Overflow
0 75 to 0 0 mm.

Plug discharge
2.5 to 0 75 mm.

To finishing-table system

One roughing Wilfley

No. 7 Anaconda classifier

Rough concentrates

Clean concentrates

Overflow, tailings   Plug, middlings

To Hancock jig system

Huntington mills

The numbers applied to the classifiers are arbitrary and are used merely for convenience. Fig. 15 shows the cross-section of the No. 2 classifier. Fig. 13 shows the No. 7 classifier, and Fig. 14 shows the distribution of products on the roughing Wilfley.

PRODUCT	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent. Total Feed	Density, Per Cent. Solids	Assay, Per Cent. Cu	Per Ct. Total Copper in Feed
Feed	. . . .	132,546	100 0	. . . .	1.39	100 0
Clean concentrates.	1,048	16,330	12.3	83.3	5.5	48 5
Rough concentrates	21,600	41,696	31.5	16.7	1 24	28.0
Plug middlings.	. . . . . .	43,594	32.9	. . . .	0 69	16 3
Overflow tailings	. . . . . .	30,926	23.3	. . . .	0.43	7.2
Total	. . . .	132,546	100 0	. . . . .	1.39	100.0

The No. 7 classifier used 127,000 gal. of fresh water, or 350 gal. per ton of feed. As seen from the sketch, Fig. 13, the classifier was a very small
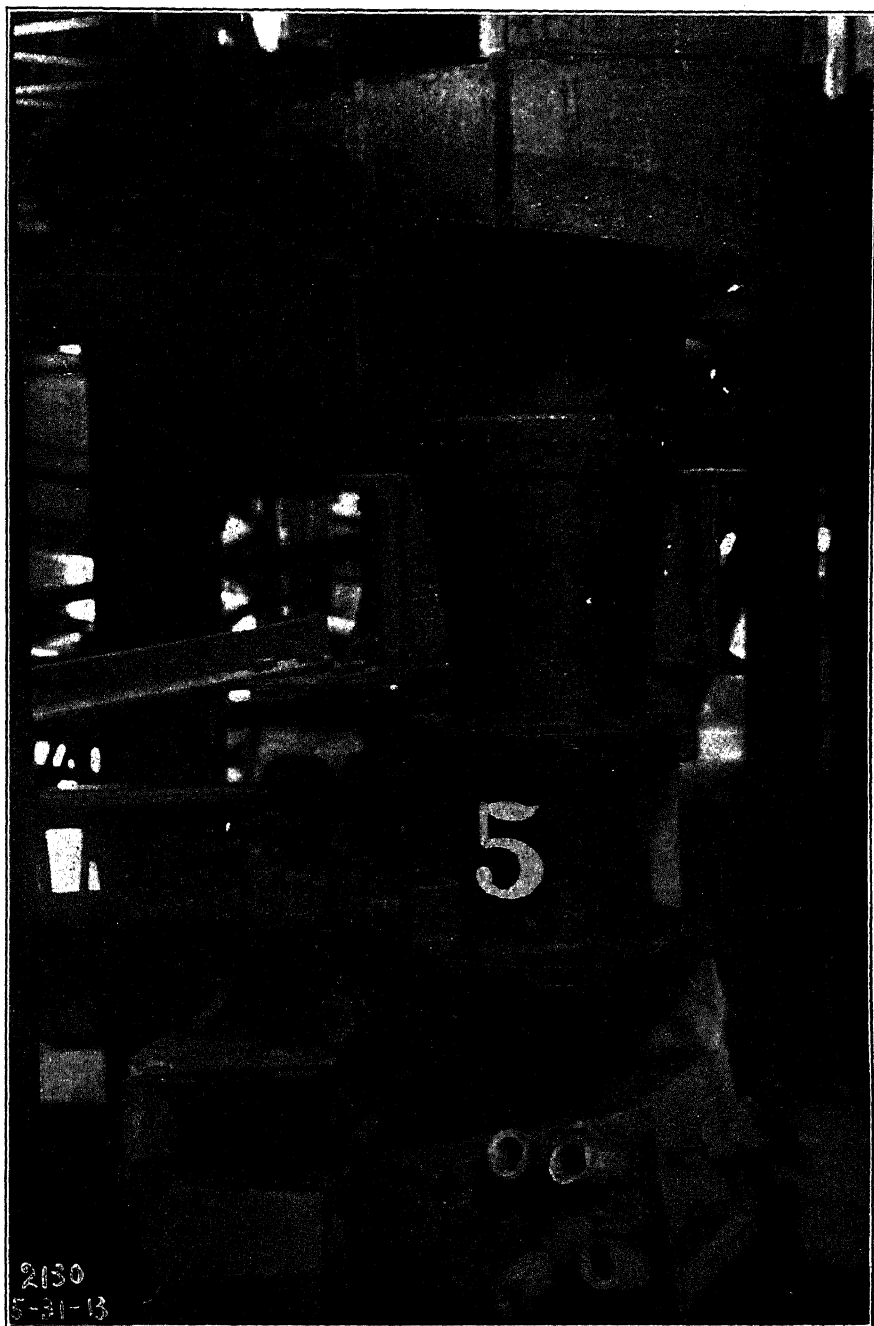
FIG. 12.—TAILINGS CLASSIFIER AT THE WASHOE MILL.

one, handling only 36 tons of feed. Still, there is every possibility of building a larger machine with just as great an efficiency. The most essential feature of this system is that the roughing Wilfley must produce a middlings-tailings product absolutely free of fine mineral. The writer has seen roughing Wilfleys at the B. & M. Concentrator doing very good work on 130 tons of 2.5 to 7.5 mm. feed.

Screen Sizing Figures.

SCREEN SIZE MILLIMETERS	PLUG MIDDLINGS		OVERFLOW TAILINGS	
	Individual Per Cent. Total Solids	Cumulative Per Cent. Total Solids	Individual Per Cent. Total Solids	Cumulative Per Cent. Total Solids
On 1.41	54.1	54.1	30.8	30 8
On 0 841	43.2	97.3	62.3	93.1
Through 0 841	2 7	100.0	6.9	100 0
Total......	100.0	100.0	100.0	100.0

Roughing Table Products—Clean and Rough Concentrates.

SCREEN SIZE MILLIMETERS	CLEAN CONCENTRATES		ROUGH CONCENTRATES	
	Individual Per Cent. Total Solids	Cumulative Per Cent. Total Solids	Individual Per Cent. Total Solids	Cumulative Per Cent. Solids
On 1.41..	2.7	2.7	3 9	3 9
On 0 841..	4.3	7.0	30.1	34.0
On 0.500	13 4	20.4	57 8	91 8
On 0.350	15 2	35.6	5.3	97 1
On 0 166.	51 9	87.5	2.9	100.0
On 0 07..	11.5	99.0	....	....
Through 0.07.	1.0	100.0	....	... .
Total	100 0	100.0	100.0	100.0

The screen sizing figures are self explanatory. The screen sizing of the rough concentrates shows the segregation of "through 0.841 on 0.500 mm." material, amounting to 57.8 per cent. of the total rough concentrates, or 18.2 per cent. of the total feed to the table. It may be well to state here that the No. 2 classifier preceding the roughing table, while doing good work, was one of the earliest fitted with the feed cone. One of the present-day cone classifiers would eliminate a great percentage of this fine ma-

terial. Some interesting tests have been made on this No. 2 classifier and the results follow. This particular No. 2 was the pioneer of the present inner feed cone classifier as used at Great Falls.
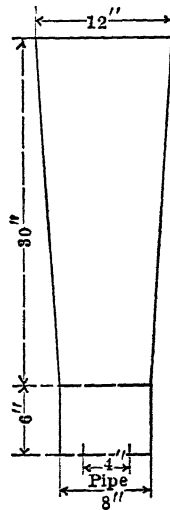
FIG. 13.—SECTION OF No. 7 CLASSIFIER.
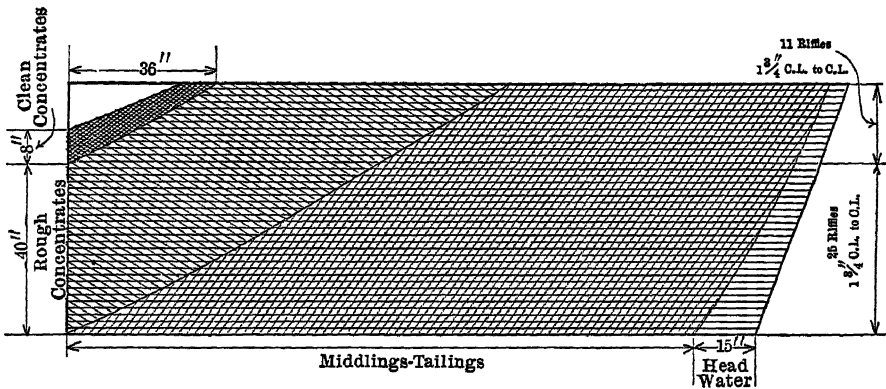
FIG. 14.—DISTRIBUTION OF PRODUCTS ON WILFLEY TABLE.

For the method of designing the Anaconda classifier reference is made to a paper before this meeting by E. S. Bardwell, The Application of Hindered Settling to Hydraulic Classifiers.[1]

8. Classifying 2.5 mm. Roll Product (Not Deslimed) in No. 2 Classifier.

The following test figures are actual practice figures. The classifier was not altered or adjusted in preparation for the test. Samples were

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[1] P. 266, this volume.

taken on two days practically at 1-hr. intervals and each sample was screen sized and sorted into free mineral grains, true middlings, and tailings grains. The classifier was fed direct from the trommel undersize, taking all of the feed at this point of a 500-ton section. The primary object of the test was to show the work of the classifier under varying
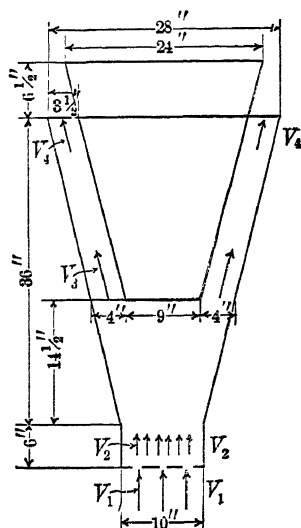
Area of top annular space	= 269.4 sq. in.
Area of bottom annular space	= 163.4 sq. in.
Area of teeter chamber	= 78.8 sq. in.
Area of constriction opening	= 33.46 sq. in.

FIG. 15.—SECTION OF No. 2 CLASSIFIER.

feed. The feed varied during this run from 90 to 250 tons per 24 hr., although the pulp volume was fairly constant. The tabulations show remarkably good classification under this changing feed.

The classifier was provided with a 1.25-in. spigot and used an average of 124,000 gal. of total fresh water per 24 hr., this being determined as an average of many samples.

The following tabulations and classification charts tell the story at the time of each different sample more clearly than words.

The symbols used in the tabulation are as follows:

V_1 = Rising velocity through constriction in millimeters per second.
V_2 = Rising velocity in teeter chamber in millimeters per second.
V_3 = Rising velocity at bottom of annular ring in millimeters per second.
V_4 = Rising velocity near top of annular ring in millimeters per second.

A = Per cent. of variation of feed pulp, gallons per 24 hr. above and below average.
B = Per cent. of variation of feed pulp solids, pounds per 24 hr. above and below average.
C = Average diameter in millimeters of quartz in plug discharge.
D = Average diameter in millimeters of mineral in plug discharge.
E = Ratio of C and D.
F = Tons of plug discharge per square inch of constriction opening per 24 hr.
G = Tons of plug discharge per square inch of cross-section of teeter chamber per 24 hr.

Details of No. 2 Classifier Tests.

TEST NUMBER	Time Taken	V_1 a	V_2 b	V_3	V_4	A	B	C	D	E	F	G
	5-14-12											
1	9.00	87	37	110	67	+1 5	+21 2	1 42	0 58	2.62	4 78	2 03
2	10.30	76	32	93	56	−17 0	−31 3	1 30	0 58	2.24	2 95	1 26
3	11:40	74	31	107	65	−1 1	−10 2	1 40	0 70	2.00	3 67	1 56
4	12·40	100	43	112	68	+1 5	+29.6	1.28	0 54	2.37	4.75	2 02
5	1:40	94	40	130	79	+18 9	+13 6	1 47	0 77	1.91	4 26	1 81
6	2 40	80	34	116	70	+3 7	−53.9	1 51	0 91	1 66	2 06	0 88
7	3:40	89	38	108	66	−0 7	+14.0	1 40	0 80	1 75	4 63	1 97
	5-15-12											
8	11:20	100	43	107	65	−4 6	+2.8	1 54	0 65	2 36	4 33	1 84
9	1:15	105	45	115	70	+2 6	+17.7	1 29	0 72	1 79	4 40	1 87
10	2:15	101	43	111	67	−2.5	−2.5	1 34	0 82	1 63	3 75	1 60
11	3:15	102	43	114	69	−2 5	−1.1	1 28	0 77	1 66	3.79	1 61
AVERAGE..	.....	92	39	111	67	0.0	0.0	1 39	0.71	1.95	3 94	1 68

a Volume of solids in plug discharge allowed for.

b Assuming the volume of solids in the teeter chamber to be equivalent to the volume passing through the constriction for the same length of column. Actually V_2 probably is nearly equal to V_1, due to the fact that the density of the pulp in the teeter chamber is greater than the density of the pulp passing through the constriction opening.

No. 2 Classifier

Test No.	Gallons per 24 Hr.	Pounds per 24 Hr.	Per Cent Total Feed	Density Per Cent Solids	Individual Per Cent. Total on Screen Sizes								Cumulative Per Cent. Total on Screen Sizes							
					2.38	1.41	0.841	0.500	0.350	0.166	0.07	Th. 0.07	2.38	1.41	0.841	0.500	0.350	0.166	0.07	Th. 0.07
SPIGOTS																				
1	106,808	319,594	67.0	28.9	2.0	35.3	39.3	17.4	2.6	2.7	0.5		2.0	37.3	76.6	94.0	96.6	99.3	99	100
2	92,208	197,510	73.1	20.0	0.4	35.7	33.1	17.7	4.8	6.7	0.8	0.3	0.4	37.1	70.2	87.9	92.7	98.9	99	100
3	96,232	245,696	69.7	25.4	1.6	40.0	34.5	15.4	3.0	3.0	0.6	0.3	1.6	43.2	77.2	93.1	95.1	99.1	99	100
4	90,166	317,952	62.3	30.8	2.0	32.2	30.4	17.0	2.4	3.9	0.6	0.3	2.0	35.8	75.2	92.8	95.2	99.1	99	100
5	90,674	285,034	63.8	27.8	4.1	43.3	34.8	10.6	2.3	2.8	0.7		4.1	47.4	82.2	91.4	97.1	99.1	100.0	
6	15,678	138,214	76.0	15.5		51.5	28.8	10.5	3.0	2.8	0.9		2.6	54.1	82.9	93.5	96.3	99.2	100.0	
7	94,092	309,830	69.1	28.8	2.4	36.3	36.2	13.7	2.8	2.8	0.6	0.2	2.4	40.7	76.9	93.3	96.3	99.2	99	100
8	97,586	280,699	71.0	28.0	4.4	37.8	29.8	18.7	3.2	3.0	0.9		4.4	50.6	80.4	93.1	90.3	99.8	100.0	
9	95,078	294,365	63.0	20.0	12.7	33.6	39.4	18.7	3.2	3.0	0.8		12.7	34.3	73.7	92.0	96.2	99.2	100.0	
10	94,000	251,251	65.0	70.4	0.9	37.1	38.5	17.1	2.6	3.0			0.7	38.0	73.5	93.4	96.2	99.2	100.0	
11	94,000	253,498	65.3	26.6	0.3	38.0	37.7	16.4	2.6	2.9	0.7		0.3	39.3	77.0	93	96	99.3	100.0	
OVERFLOWS																				
1	261,360	156,643	33.0	7.0			1.4	8.7	7.3	18.4	15.0	48.3			1.4	10.1	17.4	35.8	51.7	100.0
2	221,400	72,662	26.9	3.8				2.6	3.5	25.7	21.9	46.3				2.6	6.1	31.8	53.7	100.0
3	254,664	107,309	30.3	4.9			0.8	6.5	6.5	24.4	16.4	45.0			0.8	7.3	13.8	38.3	54.1	100.0
4	267,840	191,808	37.7	6.8				5.9	6.5	18.5	17.2	50.7				5.9	12.4	32.7	49.9	100.0
5	309,312	161,810	36.2	6.0			0.8	10.7	5.7	20.1	16.5	41.7			0.8	13.6	21.2	41.1	58.3	100.0
6	277,711	43,131	23.9	6.2				5.7	5.7	22.8	15.5	49.3				5.7	11.2	35.1	50.7	100.0
7	258,120	138,586	30.9	5.8			0.8	3.8	8.9	25.5	17.5	51.6			0.8	4.6	13.6	32.9	48.4	100.0
8	255,744	114,739	28.9	7.0			0.7	6.2	6.9	21.6	15.0	47.3			0.7	6.4	13.2	35.7	52.7	100.0
9	275,162	168,394	36.4	5.0			1.2	8.2	8.7	23.1	16.0	43.7			1.2	9.4	17.6	40.7	56.3	100.0
10	264,606	132,192	34.4	6.6			1.2	8.0	8.4	19.8	15.3	47.1			1.2	9.2	17.6	36.9	52.9	100.0
11	272,160	135,371	34.7	5.7				8.0	8.4	23.3		43.8				8.0	17.6	40.9	56.2	100.0
FEEDS					*Rising Water*															
1	244,168	476,237	100.0	30,092 / 20.2																
2	199,698	270,172	100.0	29,502 / 14.6																
3	237,973	353,005	100.0	26,491 / 15.4																
4	244,006	509,760	100.0	36,534 / 21.4																
5	285,986	446,844	100.0	34,726 / 16.5																
6	249,389	181,345	100.0	33,842 / 8.2																
7	238,812	448,416	100.0	31,708 / 19.5																
8	229,330	404,438	100.0	38,014 / 18.5																
9	246,840	462,759	100.0	40,072 / 19.5																
10	234,600	383,443	100.0	40,000 / 17.3																
11	234,600	388,867	100.0	40,110 / 17.5																

No. 2 Classifier

			SPIGOT									OVERFLOW							
			FREE MINERAL		MIDDLINGS			TAILINGS					FREE MINERAL				TAILINGS		
SCREEN SIZE, Millimeters	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size (Spigot)	Per Cent. Total Free Mineral (Feed)	Per Cent. Total Solids on Screen Size (Spigot)	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size (Spigot)	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Mineral in Overflow	Per Cent. Total Mineral in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed
						TEST No. 1													
2.38	2.0	1.3	...	...	100.0	2.0	1.5	70.0	20.0	20.3	1.4	0.5	...	...	...	100.0	...	0.3
1.410	35.3	23.7	...	10.0	24.0	8.4	6.4	90.0	20.9	28.6	8.7	2.4	1.4	0.6	0.3	98.6	1.4	2.1
0.841	39.3	26.3	3.4	14.7	...	...	...	89.2	38.5	1.0	7.3	0.4	3.8	1.2	0.2	96.6	8.1	1.7
0.500	17.4	11.7	10.8	17.1	...	...	...	66.0	15.7	1.3	18.4	6.2	4.0	4.4	2.1	95.2	7.1	3.7
0.350	2.7	1.7	33.8	8.7	...	...	...	44.0	1.2	0.1	15.9	4.9	8.7	10.4	5.1	87.0	13.9	4.7
0.166	0.5	0.3	56.0	2.9	...	...	...	33.0	0.1	...	48.3	16.0	12.7	33.8	42.9	65.0	31.4	7.7
0.07	0.2	0.2	67.0										35.0					
Th. 0.07			100.0															
TOTAL	100.0	67.0	6.2	48.8	10.4	10.4	7.9	83.4	83.4	63.0	100.0	33.0	20.2	100.0	51.2	70.8	79.8	19.4
						TEST No. 2													
2.38	3.4	1.0	...	...	100.0	1.4	1.0	88.3	29.3	21.6	2.6	0.7	...	...	...	100.0	...	0.7
1.41	35.7	26.2	1.2	5.3	98.6	35.3	26.1	81.1	15.0	11.5	3.5	0.9	...	...	...	97.3	2.6	0.9
0.841	33.1	24.2	3.7	14.0	8.4	2.8	2.1	71.5	4.7	3.5	25.7	9.5	3.5	8.6	1.0	96.5	3.4	6.4
0.500	17.1	12.9	11.7	26.8	...	...	...	70.6	0.1	3.1	21.9	12.5	21.5	18.6	10.0	78.5	16.7	7.3
0.350	4.3	3.1	28.9	23.5	...	...	...	12.6	0.1	...	46.3		41.2	77.9	41.4	58.8	27.7	7.3
0.166	6.7	4.0	29.5	3.3														
0.07	0.3	0.2	87.0	3.3														
Th. 0.07			100.0															
TOTAL	100.0	73.1	7.7	46.7	39.5	39.5	29.2	52.8	52.8	38.9	100.0	26.9	25.0	100.0	53.3	75.0	75.0	10.7
						TEST No. 3													
2.38	2.6	1.8	...	0.9	100.0	2.6	1.8	23.2	9.4	6.5	0.8	0.2	...	...	...	100.0	0.8	0.2
1.41	40.5	28.0	1.8	21.5	75.0	30.5	21.2	85.5	30.7	21.4	6.5	2.0	...	0.8	0.5	97.3	0.2	1.9
0.841	34.5	24.0	4.4	23.2	7.1	2.5	1.7	80.5	13.7	9.5	24.4	7.4	2.7	1.2	0.7	95.5	3.1	1.9
0.500	15.4	10.8	10.5	12.2	...	...	...	71.4	2.1	1.0	15.8	4.8	4.5	7.1	4.2	93.5	24.6	9.0
0.350	3.0	2.1	28.8	8.8	...	...	...	44.2	0.2	0.1	45.9	13.9	16.8	11.9	6.8	83.2	13.1	8.5
0.166	0.0	0.4	55.8	5.9	...	...	...	33.2	...	...			38.0	79.0	40.0	61.4	28.2	
0.07	0.3	0.2	66.0	3.9														
Th. 0.07			100.0															
TOTAL	100.0	69.7	7.3	41.8	35.6	35.6	24.7	57.1	57.1	40.0	100.0	30.3	22.5	100.0	58.2	77.5	77.5	23.4

No. 2 Classifier

Screen Size Millimeters	SPIGOT											OVERFLOW									
	Per Cent. Total Solids in Spigot	FREE MINERAL				MIDDLINGS			TAILINGS			Per Cent. Total Solids in Overflow	FREE MINERAL				TAILINGS			
		Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Free Mineral	Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed		Per Cent. Total Solids in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Mineral in Overflow	Per Cent. Total Mineral in Feed	Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed	

Test No. 4

Screen Size	Spigot %	Feed	Screen	Free Min	Feed	Scr Mid	Spig Mid	Feed Mid	Scr Tail	Spig Tail	Feed Tail	Over %	Feed	Screen	Over Min	Feed Min	Scr Tail	Over Tail	Feed Tail
2 38	3 6	2 2	6 5	10 5	8 0	100 0	3 6	2 2	80 3	25 7	16 0						100 0	0 8	0 3
1 41	32 2	20 0	8 3	24 1	10 1	13.2	4 3	2 7	93 4	36 2	22 5		1.0	5 9	1 6	1 0	91 1	6 5	2 4
0.841	39 4	24 0	13 6	21 8	2 0				88 4	15 2	8 5		1 2	8 8	2 0	1 2	92 2	6 0	3 4
0.500	17 6	11 5	31 5	16.1	9 7				84 5	2 0	1 3		3 0	8 0	4 5	2 8	91 2	13 5	5 1
0.350	3 9	2 4	44 0	2 8	1 2				55 5	0 3	0 2		6	17 0	8 1	5 0	82 0	13 6	1 9
0.166	0 3	0 2	50 0	6	3 6				50.0				45 6	38 0	78 1	45.6	62 0	31 6	11 9
0.07			100 0																
Th. 0 07																			
Total	100 0	62 3	10 7	100 0	41.5	7.0	7 9	4 9	81 4	81 4	50 8	37 7	58.5	24 8	100 0	75 2	75 2	28 4

Test No. 5

Screen Size	Spigot %	Feed	Screen	Free Min	Feed	Scr Mid	Spig Mid	Feed Mid	Scr Tail	Spig Tail	Feed Tail	Over %	Feed	Screen	Over Min	Feed Min	Scr Tail	Over Tail	Feed Tail
2 38	4 1	2 6	2 5	11 6	4 5	100 0	4 1	2 6	93 4	32 5	20 7		3 0	9 6	1 0	1 9	90 8	2 9	1 0
1 41	43.3	27.6	6 0	24.3	9.3	97.5	42 2	26.9	79.7	15 0	6 4		10 9	7 6	3 1	3 9	88.4	10 5	3 6
0.841	34.8	22.1	20 7	27 4	10.4				42 3	5 5	6 3		1 1	7 8	2 1	2 3	78.5	10 4	4 1
0.500	12.6	8.1	57 1	14 2	5 5				23 5	0 5	0 3		20 1	13.9	10 5	6 2	59 5	17 1	5 5
0.350	2.3	1.4	76.5	8 4	7 0				42 9	0 3	0 2		17 2	23.9	15 5	5 1	70.1	13 1	4 8
0.166	0.7	0.4	57.1	4 4	1 7								41 7	43 5	68 3	42 1	56 5	23 5	11 9
0.07																			
Th. 0 07																			
Total	100 0	63 8	9 4	100 0	38 4	97.5	40 3	29 5	44 3	44 3	28 2	36 2	61 6	26 6	100 0	73 4	75 2	26 6

Test No 6

Screen Size	Spigot %	Feed	Screen	Free Min	Feed	Scr Mid	Spig Mid	Feed Mid	Scr Tail	Spig Tail	Feed Tail	Over %	Feed	Screen	Over Min	Feed Min	Scr Tail	Over Tail	Feed Tail
2 38	2 6	2 0	2 9	23 5	9 4	100 0	2 6	2 0	90 8	26 2	20 7		0 8	3 7	0 7	0 2	100 0	0 8	0 2
1 41	51.5	39 2	4 5	20 4	8 2	97 1	50 3	38 9	88.4	16 1	6 4		5 7	7 4	1 4	1 4	96 3	5 1	1 1
0.841	28 8	22 5	11 7	12 2	8 4	4 5	1.3	0 9	59 5	9 4	0 6		5 8	17 4	2 6	4 5	92 6	9 4	3 3
0.500	11 2	8 7	35 3	17 2	8 4				57 1	1 6	0 3		22 8	30 5	10 5	3 7	82 1	17 5	4 5
0.350	2 8	1 7	40.5	6.2	2 4								16 3	43 2	15 9	11 7	56 8	10 2	1 9
0.166	2 9	2 0	42 9		2 9								49 3	30 7	69 5	41 7	28 1	28 1	6 6
0.07																			
Th. 0 07																			
Total	100 0	76 2	6 4	100 0	40 0	53 9	53 9	40.9	39 7	39 7	30 2	23 8	60 0	30 7	100 0	69 3	69 3	16 4

No. 2 Classifier

TEST No. 7

Screen Size Millimeters	SPIGOT — Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	FREE MINERAL — Per Cent. Total Solids on Screen Size	Per Cent. Total Free Mineral Spigot	Per Cent. Total Free Mineral Feed	MIDDLINGS — Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	TAILINGS — Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Per Cent. Total Solids in Overflow	OVERFLOW FREE MINERAL — Per Cent. Total Solids on Screen Size	Per Cent. Total Mineral in Overflow	Per Cent. Total Mineral in Feed	Per Cent. Total Solids in Overflow	Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Feed	TAILINGS — Per Cent. Total Solids on Screen Size	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed
2.38	4.4	3.0	4.2	14.7	7.4	100.0	4.4	3.0	45.0	16.3	11.3	0.8	7.2	1.1	0.6	0.2	100.0		0.8	0.2	
1.41	36.3	25.1	6.8	23.8	11.9	50.8	18.5	12.8	93.2	33.6	23.4	3.8	5.8	1.9	0.6	1.0	92.8		3.5	1.8	
0.841	36.5	25.4	13.0	21.0	10.8				80.0	14.2	9.1	28.5	8.8	0.7	4.8	8.7	90.0		23.8	0.7	
0.500	16.5	11.4	45.0	13.1	6.5				55.0	1.7	1.0	25.5	17.3	11.3	5.7	0.0	91.7		12.6	0.2	
0.350	3.0	2.0	2.0	18.0	9.3				20.8	0.9	0.5	15.6	35.0	76.8	38.4	0.0	82.7		33.5	0.3	
0.166	2.6	2.0	70.0		4.0							51.6				15.9	65.0			10.3	
0.07	0.2	0.4	100.0																		
Th. 0.07		0.1																			
TOTAL	100.0	69.1	10.4	100.0	49.9	22.0	22.9	15.8	66.7	66.7	46.2	100.0	23.6	100.0	50.1	30.9	76.4		76.4	23.5	

TEST No. 8

Screen Size Millimeters	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	FREE MINERAL Screen	Free Mineral Spigot	Free Mineral Feed	MIDDLINGS Screen	Middlings Spigot	Middlings Feed	TAILINGS Screen	Tailings Spigot	Tailings Feed	Solids in Overflow	OVERFLOW FM Screen	FM Overflow	FM Feed	Solids Overflow	Screen	Feed	TAILINGS Screen	Tailings Overflow	Tailings Feed
2.38	12.8	9.2	1.7	7.0	3.3	100.0	12.8	9.2	94.7	38.0	20.2	0.7	7.9	1.7	1.0	0.2	100.0		0.7	0.2	
1.41	37.0	27.1	5.0	18.9	8.2	98.3	37.0	26.5	84.1	10.4	7.2	0.6	7.0	10.9	0.2	1.7	92.1		5.6	2.6	
0.841	29.8	21.3	15.0	24.9	10.5				55.3	8.1	5.0	21.6	14.0	10.5	8.8	3.0	93.0		6.4	8.2	
0.500	13.1	9.4	44.0	15.2	10.4				27.0	1.1	0.2	17.5	24.0	70.2	8.9	3.0	66.0		18.5	6.0	
0.350	2.8	2.0	72.0	24.2	3.9				20.0	0.8	0.2	47.3	41.0		30.9	13.5	75.0		13.2	5.7	
0.166	2.9	2.0	80.0	0.2													59.0		27.8		
0.07	0.9	0.6																			
Th. 0.07																					
TOTAL	100.0	71.6	8.3	100.0	43.1	49.8	49.8	35.7	41.9	58.5	30.0	100.0	27.8	100.0	56.9	28.4	72.2		72.2	20.5	

TEST No. 9

Screen Size Millimeters	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	FREE MINERAL Screen	Free Mineral Spigot	Free Mineral Feed	MIDDLINGS Screen	Middlings Spigot	Middlings Feed	TAILINGS Screen	Tailings Spigot	Tailings Feed	Solids in Overflow	OVERFLOW FM Screen	FM Overflow	FM Feed	Solids Overflow	Screen	Feed	TAILINGS Screen	Tailings Overflow	Tailings Feed
2.38	0.7	0.4	2.7	11.1	4.8	100.0	0.7	0.4				0.2	10.8	0.5	0.4	10.7	90.0		0.7	0.4	
1.41	33.0	21.1	3.6	17.5	9.0	97.3	32.7	20.8	96.4	38.0	24.2	8.8	0.8	2.6	1.0	7.7	93.2		5.6	2.8	
0.841	39.4	25.1	12.0	27.1	11.0				88.0	13.4	10.5	8.2	11.7	12.4	7.5	7.7	93.3		6.4	7.7	
0.500	18.7	11.9	34.0	23.2	15.3				65.0	2.1	1.3	23.6	14.5	67.4	8.8	8.7	88.8		18.5	20.5	
0.350	3.2	2.0	58.0	24.0	1.6				41.1	1.5	0.9	15.7	45.0		40.8	15.9	79.0		13.7	15.0	
0.166	3.6	2.3	41.7						58.3	1.0	0.3	43.7					55.0		27.0	29.0	
0.07	0.8	0.5																			
Th. 0.07																					
TOTAL	100.0	63.6	8.1	100.0	39.4	33.4	33.4	21.2	58.5	58.5	37.2	100.0	21.7	100.0	60.6	36.4	78.3		78.3	28.5	

No. 2 Classifier

TEST No. 10

Screen Size, Millimeters	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Free Mineral: Per Cent. Total Solids on Screen Size	Free Mineral: Per Cent. Total Free Mineral Spigot	Free Mineral: Feed	Middlings: Per Cent. Total Solids on Screen Size	Middlings: Per Cent. Total Solids in Spigot	Middlings: Per Cent. Total Solids in Feed	Tailings: Per Cent. Total Solids on Screen Size	Tailings: Per Cent. Total Solids in Spigot	Tailings: Per Cent. Total Solids in Feed	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed	Overflow Free Mineral: Per Cent. Total Solids on Screen Size	Overflow Free Mineral: Per Cent. Total Mineral in Overflow	Overflow Free Mineral: Per Cent. Total Mineral in Feed	Overflow Tailings: Per Cent. Total Solids on Screen Size	Overflow Tailings: Per Cent. Total Solids on Overflow	Overflow Tailings: Per Cent. Total Solids in Feed
2.380	0.9	0.6				100.0	100.0	0.9											
1.41	37.1	24.3	3.0	13.6	5.9	97.0	36.0	23.6	94.6	36.4	23.9	0.4	1.2	5.7	0.4	0.2	93.3	1.2	0.4
0.841	38.5	25.3	5.4	25.2	10.9				87.0	15.0	9.9	2.0	8.7	7.1	2.8	1.6	92.9	7.0	2.6
0.500	17.1	11.2	12.4	25.6	11.1				65.0	1.7	1.1	8.0	7.7	5.0	2.4	1.4	93.5	7.2	4.5
0.350	2.6	1.7	35.0	11.0	4.7				41.2	1.2	0.8	6.5	19.8	11.0	11.2	6.3	88.4	17.5	5.0
0.166	3.0	2.0	58.8	20.9	9.1				63.2	0.5	0.3	16.0		9.3	7	4.1	90.7	14.5	6.0
0.07	0.8	0.5	36.8	3.7	1.6							16.3	47.1	34.0	76.0	43.1	66.0	31.4	10.8
Th. 0.07																			
TOTAL	100.0	65.6	8.3	100.0	43.3	36.9	36.9	24.2	54.8	54.8	35.0	34.4	100.0	20.6	100.0	56.7	79.4	79.4	27.3

TEST No. 11

Screen Size, Millimeters	Per Cent. Total Solids in Spigot	Per Cent. Total Solids in Feed	Free Mineral: Per Cent. Total Solids on Screen Size	Free Mineral: Per Cent. Total Free Mineral Spigot	Free Mineral: Feed	Middlings: Per Cent. Total Solids on Screen Size	Middlings: Per Cent. Total Solids in Spigot	Middlings: Per Cent. Total Solids in Feed	Tailings: Per Cent. Total Solids on Screen Size	Tailings: Per Cent. Total Solids in Spigot	Tailings: Per Cent. Total Solids in Feed	Per Cent. Total Solids in Overflow	Per Cent. Total Solids in Feed	Overflow Free Mineral: Per Cent. Total Solids on Screen Size	Overflow Free Mineral: Per Cent. Total Mineral in Overflow	Overflow Free Mineral: Per Cent. Total Mineral in Feed	Overflow Tailings: Per Cent. Total Solids on Screen Size	Overflow Tailings: Per Cent. Total Solids on Overflow	Overflow Tailings: Per Cent. Total Solids in Feed
2.380	1.3	0.8	3.5	12.8	5.6	100.0	36.5	0.8											
1.410	38.0	24.7	5.5	19.8	8.6	96.5		24.0	94.5	35.6	23.2	0.4	1.2	18.2	0.9	0.5	81.8	1.7	0.4
0.841	37.7	24.6	17.0	27.4	12.2				82.3	13.0	8.4	2.0	8.0	4.3	1.4	0.8	95.7	7.0	2.7
0.500	16.4	10.7	50.0	14.5	6.4				50.5	1.0	0.4	8.9	4.4	15.0	5.0	2.8	84.0	7.5	5.0
0.350	3.0	2.0	76.5	7.5	9.2				24.5	0.7	0.3	1.3	23.3	25.5	23.1	13.0	75.0	17.5	4.9
0.166	2.9	1.9	63.6	4.1	1.7				36.4	0.3		5.3	15.3	7	4.5	2.6	92.5	14.2	4.9
0.070	0.7	0.5										15.2	43.8	37	05.1	36.6	62	27.3	9.5
Th. 0.070																			
TOTAL	100.0	65.3	10.4	100.0	43.7	37.8	37.8	24.8	51.8	51.8	33.7	34.7	100.0	25.3	100.0	56.3	74.7	74.7	26.0

PER CENT. OF SOLIDS EACH SIZE

Screen Size, mm.

LEGEND

⬚ Middlings in plug   ☐ Tailings in plug   ⬚ Free Mineral in plug

⬚ Tailings in Overflow   ⬚ Free Mineral in Overflow

The entire area represents feed to classifier. Free mineral consists of 15 per cent. insoluble concentrates. Heavy black line separates plug and overflow. Through 0.07 free mineral consists of 60 per cent. insoluble concentrates.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 1.

PER CENT. OF TOTAL SOLIDS EACH SIZE

100

80

60

40

20

0

Screen Size, mm.

2.50 2.38   1.41   0.841   0.500 0.350   0.166 0.070   0.000

0   20   40   60   80   100

LEGEND

Middlings in Spigot   Tailings in Spigot   Free Mineral in Spigot

Tailings in Overflow   Free Mineral in Overflow
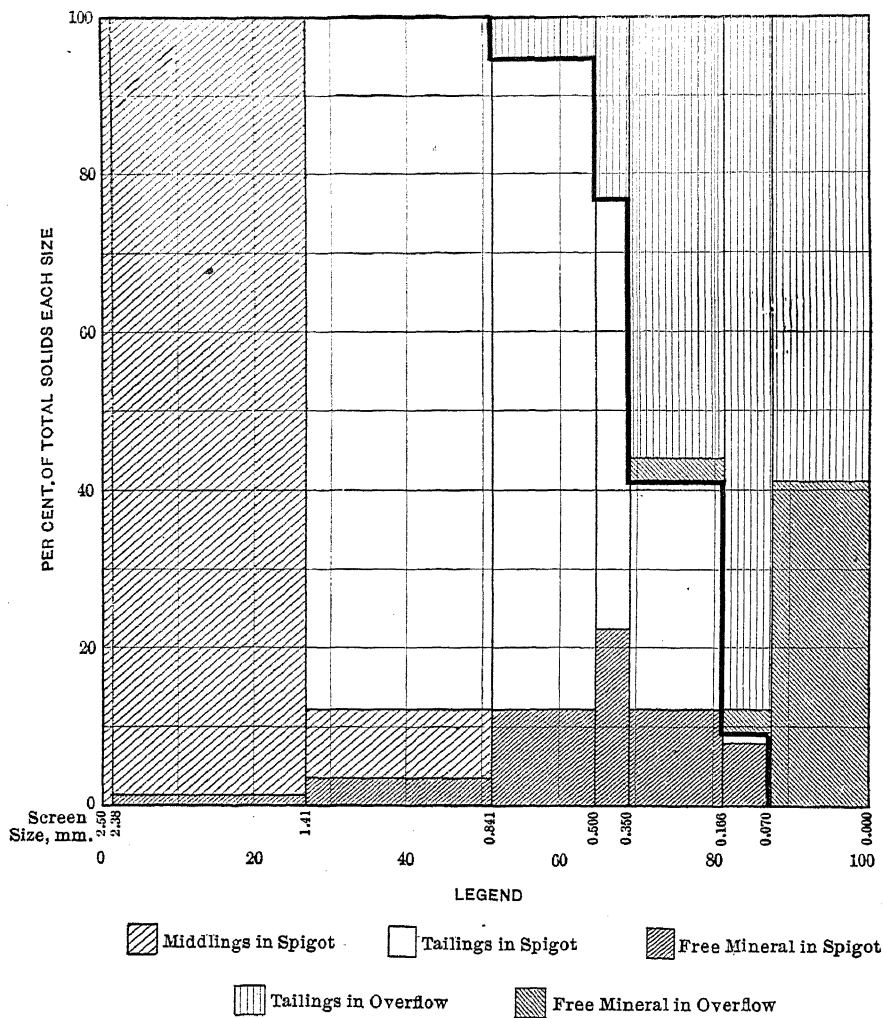
The entire area represents the total feed to classifier. Middlings on 2.38 all true middlings. Middlings on 1.41 consist of free mineral and tailings. All free mineral on 0.07 mm. consists of 15 per cent. insoluble concentrates. All free mineral through 0.07 mm. consists of 60 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 2.
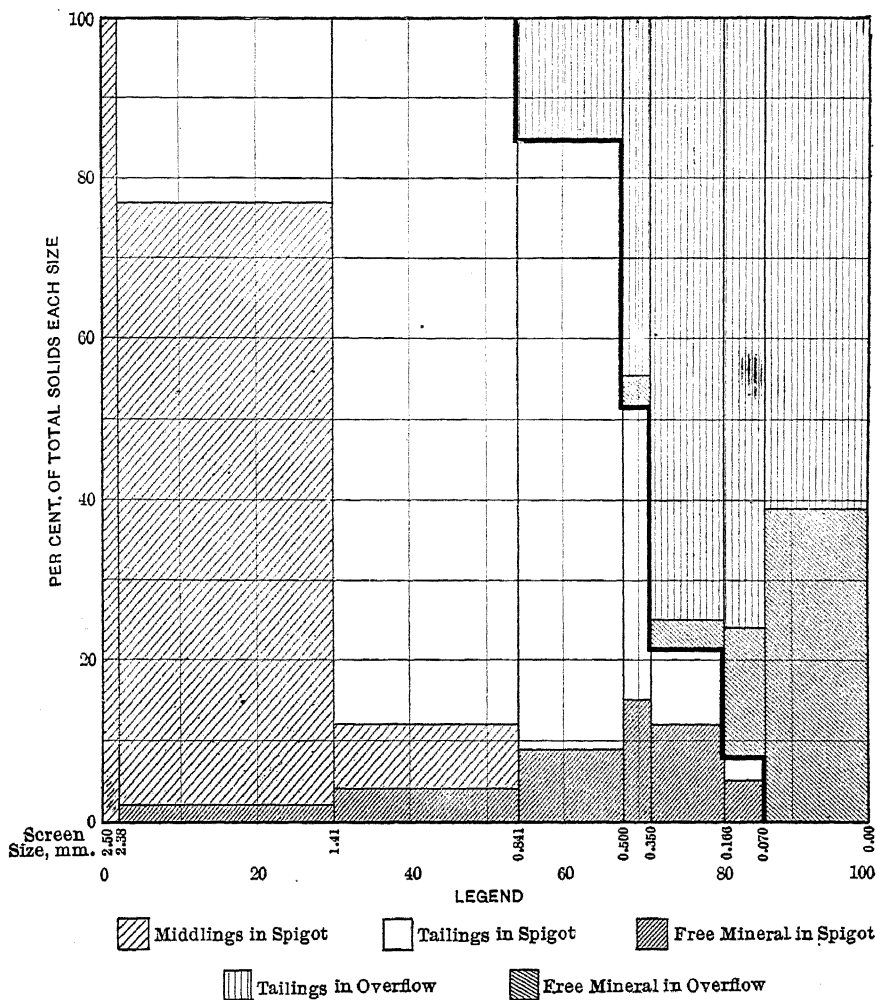
The entire area represents the total feed to classifier. Middlings on 2.38 mm. all true middlings. Middlings on 1.41 and 0.841 contained considerable free mineral. Free mineral = 15 per cent. insoluble for all sizes on 0.07 mm. Free mineral through 0.07 mm. contains 60 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 3.
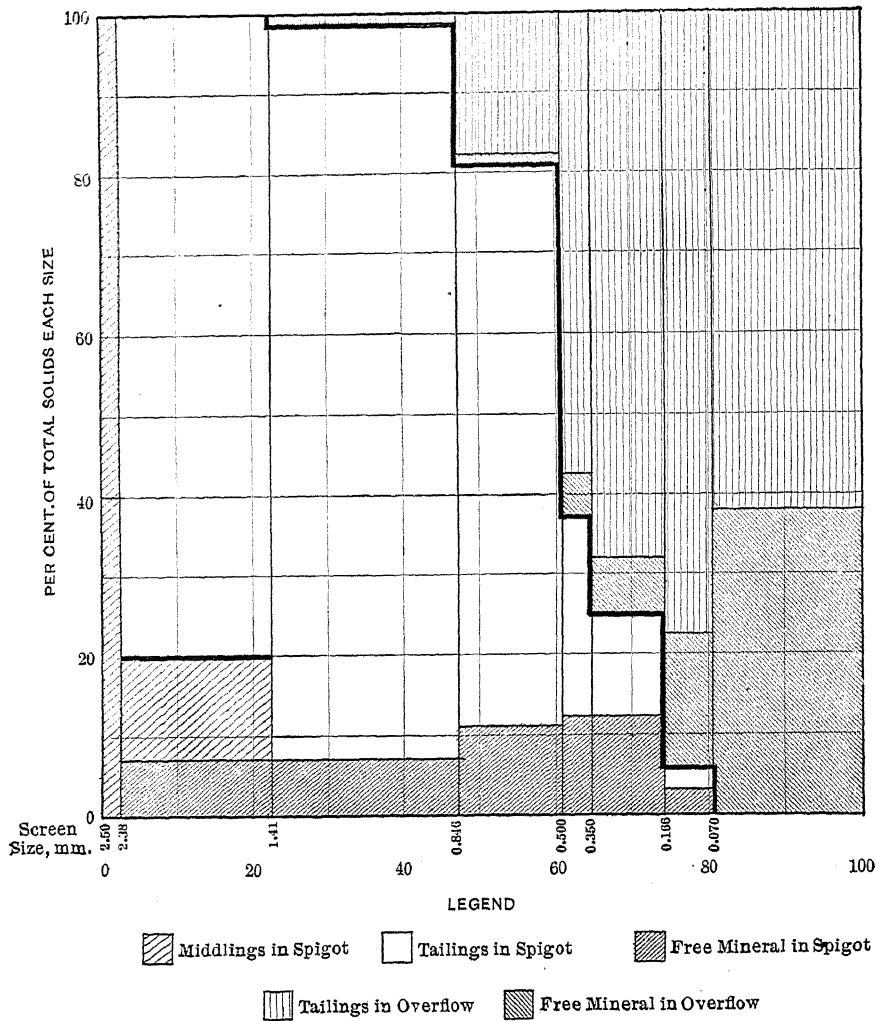
LEGEND

▨ Middlings in Spigot   ☐ Tailings in Spigot   ▨ Free Mineral in Spigot

‖ Tailings in Overflow   ▨ Free Mineral in Overflow

The entire area represents the total feed to classifier. All middlings true middlings Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 4.
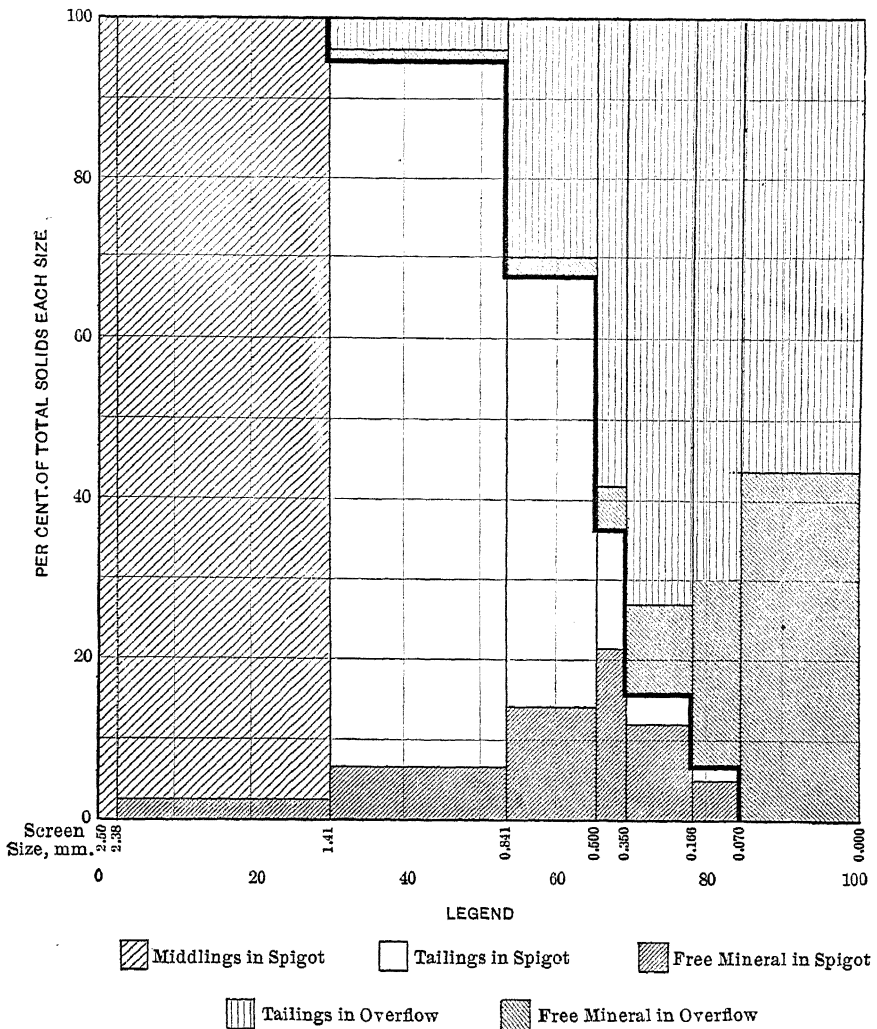
The entire area represents the total feed to classifier. Middlings contain some tailings material. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 5.
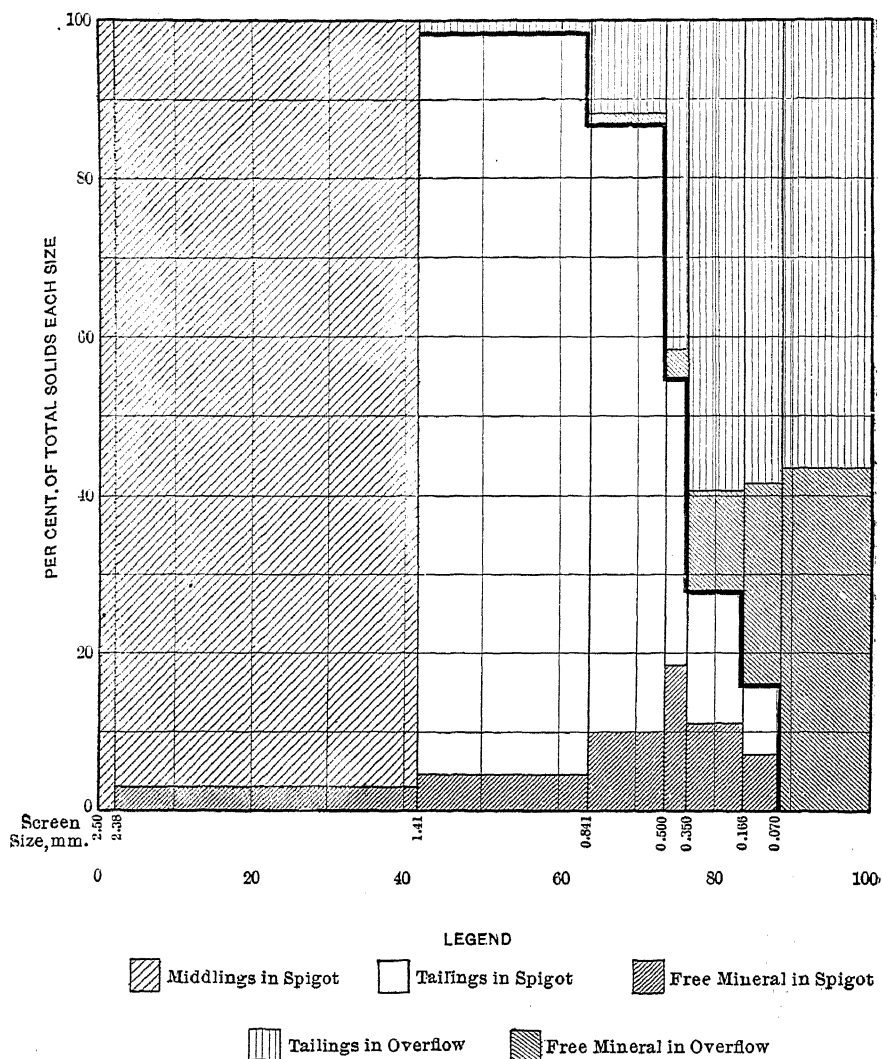
The entire area represents the total feed to classifier. Middlings considerable tailings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 6.
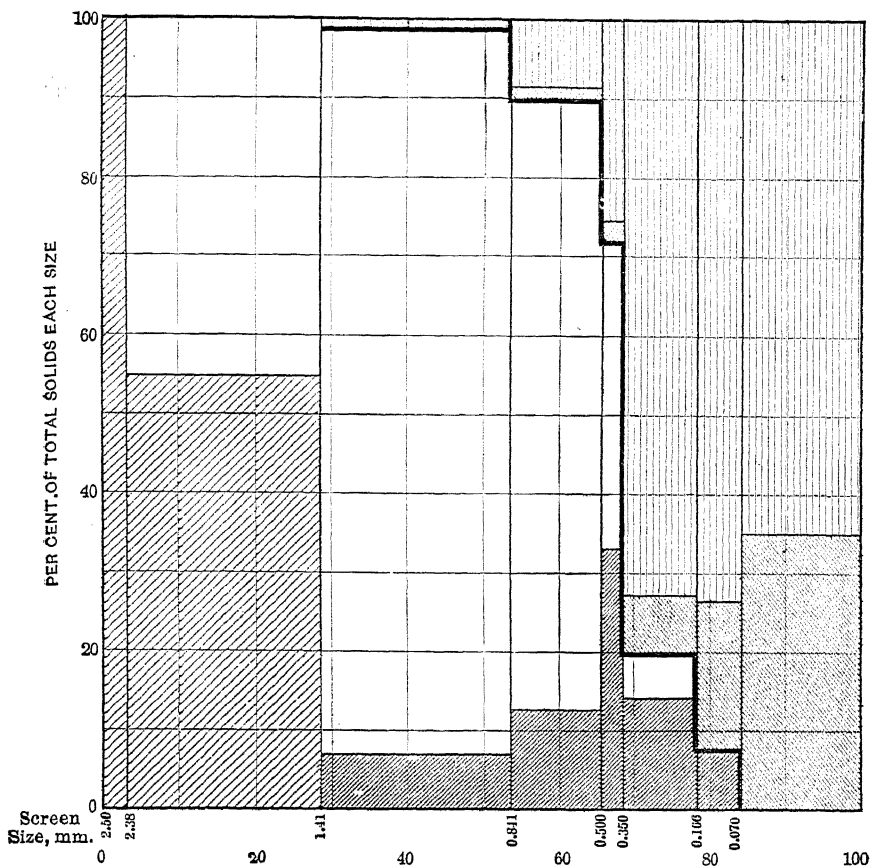
LEGEND
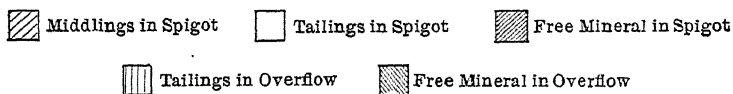
▨ Middlings in Spigot   ☐ Tailings in Spigot   ▨ Free Mineral in Spigot

▥ Tailings in Overflow   ▨ Free Mineral in Overflow

The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. imsoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 7.
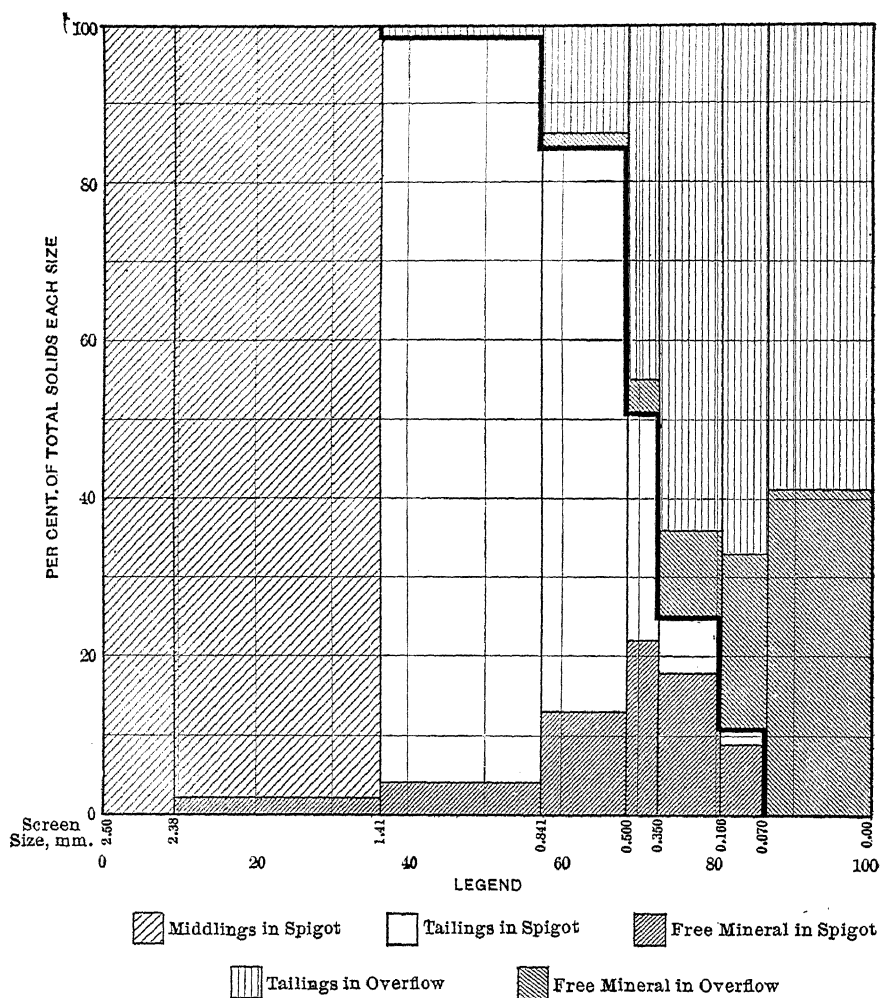
The entire area represents the total feed to classifier. Middlings contained some tailings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER.   TEST No. 8.
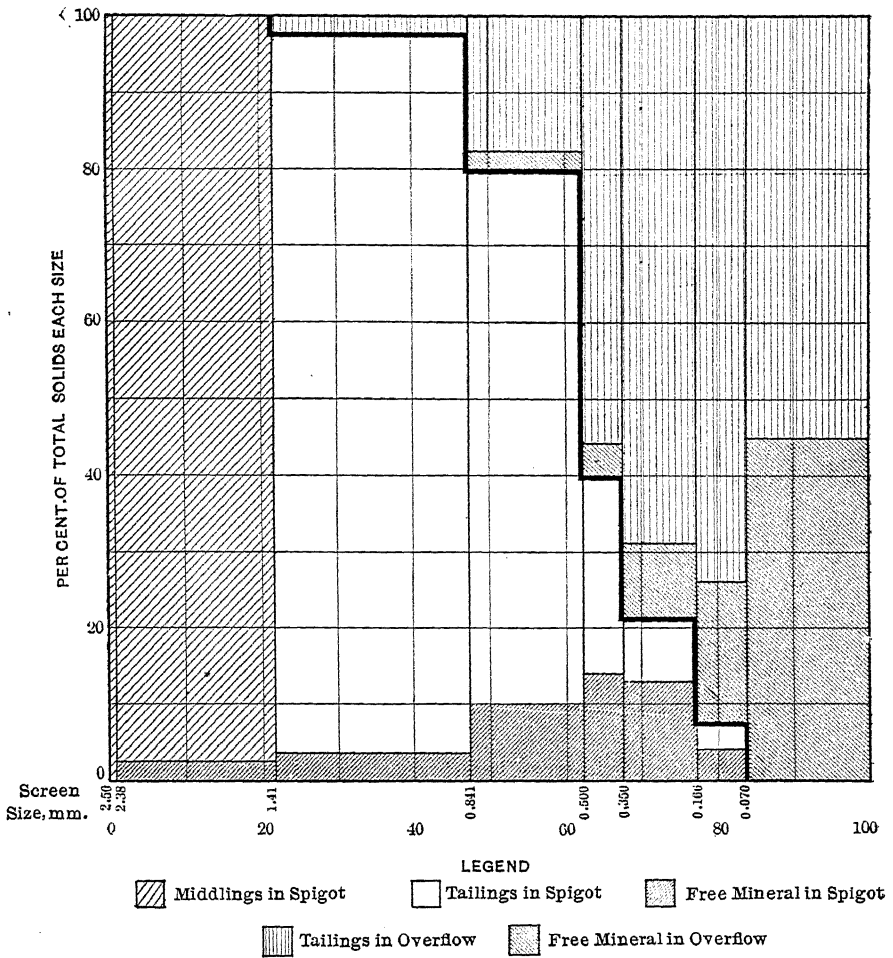
LEGEND

▨ Middlings in Spigot ☐ Tailings in Spigot ▦ Free Mineral in Spigot

▥ Tailings in Overflow ▩ Free Mineral in Overflow

The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 9.

The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 10.

LEGEND

///// Middlings in Spigot   ☐ Tailings in Spigot   ▨ Free Mineral in Spigot

||||| Tailings in Overflow   ▨ Free Mineral in Overflow
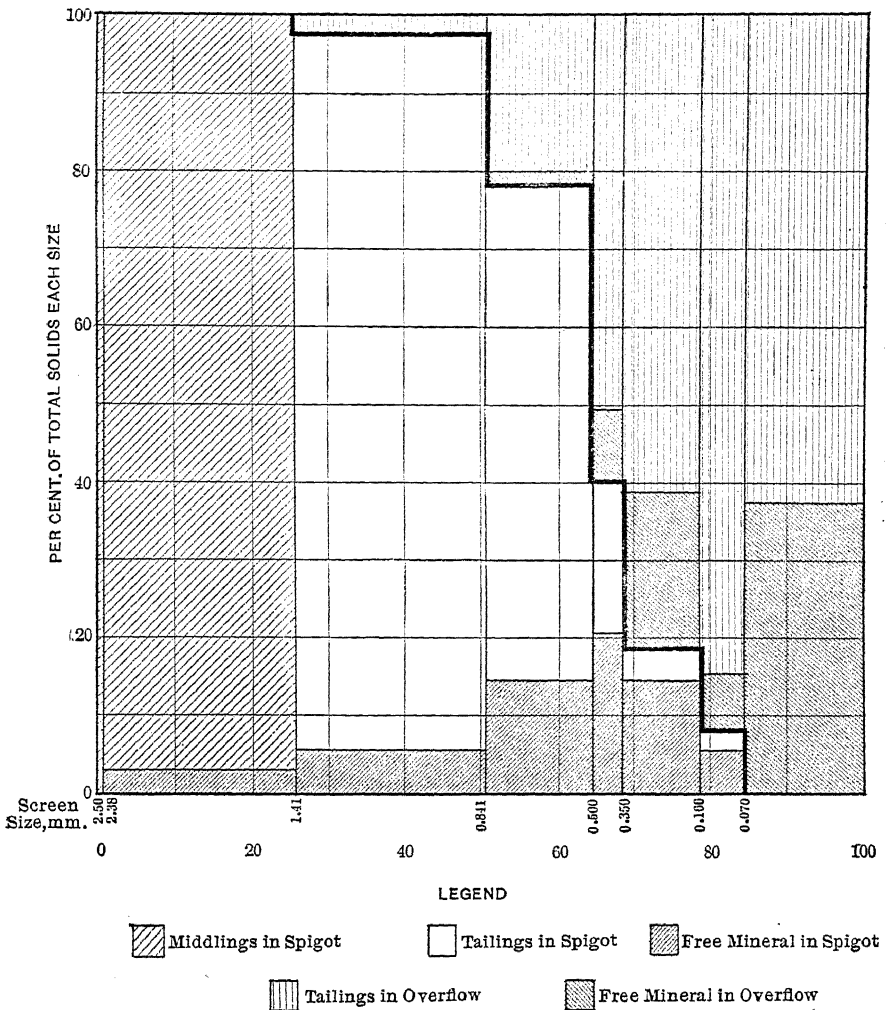
The entire area represents the total feed to classifier. Middlings all true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 11.

Particular attention is called to the consistency of classification, and especially the low water consumption.

Richards's velocity[1] for full teeter of 1.37 mm. grains is 53 mm. per second. That obtained in the classifier was 39, but this calculation was made assuming the same density in the teeter chamber as in the sorting column. As a matter of fact, had this density been determined, the rising velocity in the teeter chamber would have checked Richards's velocity. The low consumption of water is due to a great extent to the inner cone.

9. The Inner Cone.

The inner cone, in the writer's opinion, is a great improvement over the type of classifier without the cone. The chief function of the cone is to produce a constant rising current under all conditions of operation. It also acts as a feed box and reduces liability of choke-ups. By utilizing every drop of feed water for the production of the rising current, the amount of rising fresh water is not much greater than that required to produce the quicksand or full teeter condition in the teeter chamber. This fact not only reduces considerably the total fresh water consumed, but produces a much better average classification than a classifier without the cone. In a classifier with no inner cone, if the solids in the feed pulp decrease the teeter chamber may disappear entirely and we have simply a free-settling classifier with considerable fine material being discharged through the spigot. But if in an inner cone classifier the density decreases, i. e., the volume remains constant but the solids decrease, we still have the same rising current, which has a tendency to lift out all of the fine material and a good part of the coarser feed. The inner cone has given best results when so designed that the velocity at the top of the annular space is the same as the free-settling velocity of the maximum quartz grain we wish to overflow and the velocity at the bottom of the annular space is approximately 25 per cent. greater. Some particles are drawn up into the space which eventually settle down and are discharged through the spigot.

As seen in Fig. 10, the overflow is constantly under a head of from 5 to 6 in. in the inner cone. Any tendency to bank, results in the increase of this head and a cleaning up of the banking material. However, banking or choking is rare with the inner cone type.

The advantages of the inner cone may be summed up as follows:
1. A uniform close classification is obtained.
2. The water consumption is decreased.
3. The liability of choke-ups is decreased.

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[1] *Ore Dressing*, vol. iii, p. 1462 (1909).

4. Boiling in the overflow is prevented, thereby preventing coarse material being thrown over.

The chief disadvantage is the requirement of 8 in. of extra head room.

In any classifier or classifier system, the most important demands to be satisfied are: (1), flexibility of each unit and of the classifier system as a whole; (2), maximum tonnage; and (3), minimum water consumption. Let us see how the Anaconda classifier answers these fundamental requirements.

(1). The Anaconda classifier system is remarkably adapted to meet any given set of mill conditions. The classifier units can be operated under the direct system or under the indirect system. In the direct system the slime is the product of the last classification. In the indirect system the slime is the first product removed. The direct system is best adapted to conditions where mill head room is exceedingly valuable. The total pulp, usually a screen undersize, is fed directly into a table-feed classifier which overflows all material from a given size, usually 0.75 mm., down to 0.0 mm.

The overflow is easily laundered or pumped to some point where it can be further classified into slime and one or more classified table feeds. The valuable point here is that the second and third classifiers in this series can be placed immediately preceding the particular concentrating machine best adapted for the treatment of each product, allowing the practice of running very dense spigots as feed to the concentrators, which are usually some type of shaking table. Reference to Fig. 8 will show the manner of separation of such a feed into concentrates, middlings, and tailings by a Wilfley table. The concentrates are very clean, usually containing from 9 to 15 per cent. of gangue; the middlings are very small in tonnage, while the tailings are very dense and assay usually from 0.30 to 0.40 per cent. of Cu.

The indirect system first removes the slime (98 per cent. through 200 mesh), which can be segregated by itself in a plant equipped for slime treatment. The density of the slime produced in the indirect system is from 160 to 200 per cent. as dense as in the direct system. This is due to the fact that no hydraulic rising water is required for the separation of the slime alone. The spigot of the deslimer is fed directly into a table feed classifier, which affords to the classifier an absolutely constant volume of feed pulp. This fact is the most important feature of the indirect system. A variable volume of feed pulp to a deslimer is immaterial provided the deslimer is designed to handle the maximum volume. A variable feed volume to a table-feed classifier is serious in that the rising velocity varies proportionately with the feed volume.

As to the flexibility of each unit, the tabulation following Fig. 10 shows

the results of a variable tonnage. This particular unit, which is one of the earliest inner cone types, received as its feed the total undersize of two 2.5 mm. round-hole trommels. This feed was far from constant, varying from 90 to 255 tons per 24 hr., and yet the classifier maintained practically the same degree of classification under all conditions. A variation of 40 per cent. of its rated tonnage, either way, is not serious in an Anaconda classifier. The hydraulic water is the only adjustment necessary, and this is only a slight one.

The effect on the degree of classification due to variations of the percentage of fine material (mine fines) in the feed is minimized in the Anaconda classifier. In any mill which is not equipped with a central crushing plant the variations in the percentage of mine fines in the original ore and even the percentages of fines due to crushing are considerable. To perform consistent work a classifier must be able to absorb these fluctuations in the quality of its feed. On pp. 296 and 297 are shown two complete sets of figures illustrating to a certain extent how well this function is performed by the classifier.

A comparison of these two sets of figures shows the same quality of work on the rich fine feed as upon the coarser and more impoverished feed. Both sets of figures were obtained under actual operating conditions.

(2). The maximum tonnage of the classifier is limited only by practice. In Montana it is the custom to design each unit to treat from 150 to 350 tons per 24 hr., depending upon conditions of practice. Table-feed classifiers seldom treat less than 250 tons, while deslimers working on very fine feed, 2.0 mm. or under, will average from 180 to 250 tons. Deslimers treating material whose limiting size is larger than 2.0 mm. have a capacity of from 250 to 400 tons per 24 hr. The large tonnage handled by a single unit is one reason for its great flexibility.

(3). The consumption of water by these classifiers is remarkably low. In all desliming Anaconda classifiers practically no rising hydraulic water is used, leaving only that which is added in the spigot to be charged against the classifier. However, the addition of water in the spigot reduces the consumption of water in the subsequent classifier. The deslimed overflow of the table-feed classifier is dewatered, and the water thus removed is used as hydraulic water in other classifier units of the system or as dressing water for concentrating machines. Thus the total water consumption for a system of classifiers is less than is shown by the sum of the individual units. The average water consumption in practice is as follows:

Desliming classifiers:—600 gal. per ton of feed.
Table-feed classifiers:—450 to 900 gal. per ton of feed.
The relatively large consumption of water by the deslimer is due to

operating with a spigot less dense than is used in table-feed classifiers.

Fulfilling as it does the requirements of flexibility, efficiency of classification, high tonnage, and minimum water consumption, together with simplicity of construction, the Anaconda classifier is a great stride in the progress on modern ore concentration.

The writer wishes to thank A. E. Wiggin for his assistance in obtaining data from the Washoe Concentrator, and A. E. Wheeler and C. W. Goodale for reading and criticizing the paper.

Ore-Dressing Improvements.

BY ROBERT H. RICHARDS, BOSTON, MASS.

Introduction.

WALTER RENTON INGALLS recently gave a very interesting talk before the student mining society of the Massachusetts Institute of Technology. In it he showed the present status of mining as he sees it. In the talk he explained that, while discoveries of great mining districts are still possible, they are being made less and less often; that the great mining companies, although there are large numbers of mines offered for sale, are finding it less and less possible to find new mines that are up to the standard of richness that they wish for purchase.

The natural conclusion is that better, cheaper, and more efficient methods of mining, concentrating, and smelting are needed if the lower-grade mines are to be worked at a profit. This is true also if the high-grade mines are to yield their greatest profit.

This paper deals only with the concentrating or ore-dressing problem, and gives some thoughts which point towards greater profit.

In a paper on The Development of Hindered-Settling Apparatus the writer showed that there were two main lines of work that would lead to greater saving, lower tailing assay, and, therefore, greater profit.

The first is the recrushing of middling or tailing products for the saving of the contained values. This is well understood and largely carried out in existing mills.

The second is the saving of fine free mineral at the first opportunity. This is a matter less well understood in the mills, and to dwell upon this the present paper is written.

The writer is himself convinced and hopes to convince the reader that, for obtaining the above result, not only is better classification needed, but also more classification; that is, there should be more spigots or pockets to the classifier. His attention was recently called to a small mill which had a three-pocket cone classifier fed with from 2 to 0 mm. stuff, supplying feed to four Wilfley tables. The first spigot had about all the

sand and a great amount of slime, the second and third spigots and the overflow were about all slime. As a result, all four Wilfley tables had wide bands of slime on them and all their tailing products were charged full of fine free mineral. In fact, the cone classifier was not classifying the ore at all, and the mill process was greatly harmed from the lack of classification. It is a proved fact that when the feed to a Wilfley table carries slime, then the Wilfley tailing will carry fine free mineral. When this ore was tested in the laboratory it easily yielded 12 spigots, all of which when fed to Wilfley tables gave no slime bands on the table, and the fine free mineral was absent from the tailing of all the coarser Wilfley tables and present only in small quantity in a few of the finer.

All classifiers which classify by a rising current of water have a feature which we call the sorting column, in which the sand is trying to settle, and is allowed to settle or is prevented from settling, according to the size and weight of the grains, by the velocity of the rising current of water, a stronger current lifting larger and heavier grains, a weaker current lifting smaller and lighter grains. This sorting column, whether square or cylindrical in shape, will have adverse downward currents of water carrying slime down into the spigot, thus defeating the whole object of classification unless some special provision is made to stop it.

To obtain better classification, and prevent the adverse down currents two methods have been employed:

1. Causing a vortex or horizontal rotation in the sorting column.
2. Using a pulsating rising current.

Both of these methods give better classification by destroying the hurtful downward current, avoiding the carrying of slime into the spigot, and preventing fine free mineral from going into the tailing of all the coarser Wilfley tables.

To obtain more classification we simply lengthen the classifier, using more pockets. As the benefit of this may not be apparent to the concentrator man, figures are here given to prove the point:

Three-Spigot Classifier.

Spigot.	Quartz. Diameter, Millimeters.	$\frac{1}{4.5}$ of the quartz is galena Diameter, Millimeters.
1................	6.3 to 2.15	6 3 to 0.614
2..........	2.15 to 0.73	0.614 to 0.208
3..........	0.73 to 0 25	0.208 to 0 0715
Overflow....	0.25 to 0.00	0.0715 to 0 0000

Twelve-Spigot Classifier.

Spigot.	Quartz. Diameter, Millimeters.	$\frac{1}{3.5}$ of the quartz is galena Diameter, Millimeters.
1	6.30 to 4 79	6 30 to 1.37
2..	4 79 to 3.66	1 37 to 1.05
3.	3.66 to 2.80	1 05 to 0.80
4	2 80 to 2.14	0.80 to 0.61
5....	2 14 to 1.63	0.61 to 0 47
6	1.63 to 1.26	0.47 to 0.36
7....	1.26 to 0 96	0 36 to 0.27
8	0.96 to 0 73	0 27 to 0 21
9	0.73 to 0 56	0 21 to 0 16
10	0.56 to 0.43	0 16 to 0.12
11	0.43 to 0 33	0 12 to 0 094
12	0.33 to 0 25	0.094 to 0 071
Overflow...	0.25 to 0.00	0.071 to 0 000

We will take two instances: 1, the classifier which has only three pockets (see table) and is classifying stuff from 0.25 in. (6.3 mm.) to 0; 2, the classifier with 12 pockets, treating the same product. We will suppose that the classification is being done upon quartz (sp. gr. 2.64) and galena (sp. gr. 7.5) ore, and we will consider it to be a free-settling classifier, which will demonstrate the point better than if hindered settling was used. It will be borne in mind that with free settling the classified products after the first spigot and before the final overflow will contain grains where the quartz is approximately 3.5 times the galena in diameter.

If we consider any pair of products except the first, which in both instances has the added increment in it (see paper on The Development of Hindered-Settling Apparatus), and the overflow, both of which require special treatment, different from that of all the intermediate spigots, we believe it will be clear that a Wilfley table treating the second spigot of the three-spigot classifier, which has to separate 0.614 to 0.208 mm. galena from 2.15 to 0.73 mm. quartz, will not have so easy a task as the Wilfley table which treats the second spigot of the 12-spigot classifier and has to separate 1.34 to 1.001 mm. galena from 4.70 to 3.51 mm. quartz. The same opinion will hold if the third spigot of the three-spigot classifier is compared with the twelfth spigot of the 12-spigot classifier, and is true of all the intermediate spigots of the 12-spigot classifier. The comparison would be much stronger if quartz, 2.65 sp. gr., and pyrite, 5 sp. gr., were used, and still more so if quartz, 2.65 sp. gr., and blende, 4 sp. gr., were used.

To put the case still more strongly the writer adds the following facts:

The late George Woodbury, of California, found that the Wilfley table could not compete with the vanner in treating unclassified feed of the gold mills of the mother lode of Calitornia which were treating 100 tons a day, more or less. He did not locate the cause of this difference between the machines, or, if he did, he did not make his views public. At the Massachusetts Institute of Technology we have located the cause. The chief loss in the tailing of the Wilfley or vanner, when treating unclassified feed, is the presence of fine free mineral. The maximum size of the grains of fine free mineral in the tailing of the Wilfley table is larger than that of the vanner. This amply explains Woodbury's observation. We have many measurements on these sizes, but want better check results before we publish them.

Next, let us see what a long classifier with many spigots will do, which a short classifier cannot do. We shall find, as we follow on down from the coarser spigots toward the finer, that Wilfley tables will yield tailing products free from fine free mineral and, therefore, at the minimum assay that can be obtained without further re-grinding. After a time, however, we shall arrive at a spigot where the heavy mineral in the spigot is the same size as the normal size of Wilfley fine free mineral. Then the Wilfley table breaks down, can no longer yield clean tailing, and the vanner should take its place, because the fine free mineral in the vanner tailing is smaller than that in the Wilfley table. If we follow still further down the spigots we shall find a spigot or the final overflow where the size of the heavy mineral is the same as the size of the fine free mineral in the vanner tailing. Then the vanner breaks down, can no longer yield clean tailing, and we go to the new great Anaconda round table, which takes all the rest down to the size of the heavy mineral that floats away in a slow-moving stream of water. With this the methods of water-gravity separation of fine material, that are now in sight, close.

The critic will say: "But you are diluting your slime beyond reasonable limits by using so many pockets." The writer answers that that depends upon whether the classifier is evenly fed. If it is not, the water may be unreasonably large in quantity, but if devices can be installed that give even, constant feed, as for instance, those in the Utah porphyry mills, then there need be no fear of excess water. The following argument will show this: Suppose we have to classify 400 tons in 24 hr., and we wish to classify into 15 spigots. It takes 3.5 tons of water to 1 ton of sand for feeding the classifier; again 3.5 tons of water rising in a sorting column to allow 1 ton of sand to go down to the spigot, whether coarse or fine; again 3.5 tons of water to 1 ton of sand to discharge it through the spigot. Next, suppose we need 350 tons per 24 hr. from the spigots and 50 from the overflow. What difference does it make whether we settle the 350 tons of sand in one spigot against (multiplying by 3.5) 1,225 tons of rising

water, or settle the 350 tons of sand divided out between 15 sorting columns, requiring 1,225 tons of rising water also divided out between the 15 sorting columns. The 15-spigot classifier, then, if it is steadily and evenly fed, requires no more water and delivers overflow no more dilute than a one-spigot classifier.

The writer believes he has clearly proved that both better classification and more classification are needed.

Let us see what difficulties, if any, have to be overcome in order to put good classifiers into a mill. 1. No one has yet succeeded in designing a good classifier that does not require some attention. The concentrator man who swears by fool-proof machinery and declares that nothing shall go into a mill that is not absolutely fool-proof can never get good classification or make the highest saving. 2. There is a certain lack of breadth about concentrator men who, naturally enough, perhaps, like to have their own machines, and refuse to allow the machines of outsiders to succeed in their mills. One instance of this may be mentioned. A classifier which had a screen in it was tested in a mill. In some way unknown to the designer the screen had holes develop in it as large as a crowbar would make. The manager asked the concentrator man to have samples taken and assays made to see what results the classifier would give. Of course the result was as bad as could be and the classifier was condemned and thrown on the junk pile. If the holes came by accident, the classifier was not in fit condition to test. This paragraph really belongs to the discussion of professional ethics, which has been so much under discussion lately. For example, has a concentrator man a right to turn down good machinery by unfair tests and deprive the stockholders of his company, whose money pays his salary, of the extra dividend that might be developed?

If, then, it be admitted that fool-proof machinery of certain classes is bad machinery and that more perfect performance can only be obtained where the machine requires greater care, how can this care be secured? The answer is, get a higher-priced man to inspect and see that the machines are run properly and are always in good order.

Good classification can save large sums of money, $25,000, $50,000, $100,000 or more a year. If to run such classifiers we require a $1,500 man and three of him for the three shifts, it would pay well to spend $4,500 a year to make any of the above figures of saving; $4,500 is only 18 per cent. of $25,000 and 9 per cent. of $50,000. What successful business man hesitates to spend 25 or even 50 per cent. of the net income in advertising in order to make the business? It is common, every-day custom. Then why should we not spend 18 or 9 per cent. of the increased net income in order to bring ore dressing to the highest pitch of profit?

The conclusion, then, is that we need better classification, more classification, and higher-priced, broader-minded men to see that the greatest benefit is derived from the improved machinery.

DISCUSSION.

C. D. DEMOND, Anaconda, Mont. (communication to the Secretary*):—Any one who has observed the increased efficiency of the Wilfley tables in a mill when fed from a reasonably good hydraulic classifier, as compared with tables fed from a *spitzkasten*, will agree to the advantage of "better classification." It must be gratifying to Dr. Richards to know that, directly or indirectly, many mill men are responding to his persistent advocacy of correct principles of ore dressing. As he says, these principles mean much for both the conservation of resources and the profit of operators.

When Dr. Richards gives us accurate figures as to the maximum size of free mineral grains in the tailings of the Wilfley table and of the vanner, they will be of a good deal of practical value in delimiting the proper fields for the Wilfley, the vanner, and the round table. He has coined the terms "free settling" and "hindered settling."[1] The series may be completed by adding "crowded settling," the main principle of action on both the vanner and the Wilfley table, which is the principle of separation in a mass of grains crowded as closely together as they can be and yet continue free to move between one another; and the occurrence of larger mineral particles in the Wilfley tailing than in that from the vanner is apparently caused by the more violent agitation, due chiefly to the purposely sudden reversal of the stroke. It seems likely that the agitation caused by the wash water dropping directly upon the concentrate on the vanner is more harmful than that due to the shake. For the very finest material that can be concentrated by usual methods, the round table is superior to the vanner because it does not have even the gentle agitation of the latter. The action on round tables is, not "crowded settling," but "film sizing,"[2] but agitation is harmful in the latter action. George Gates proved this harm 20 years ago: in making the first successful large-scale application of canvas tables in California gold mills he abandoned all shaking motion, although he had patented canvas tables to which such motion was applied.

I hope that Dr. Richards's results as to free mineral in tailings will cover different conditions as to violence of agitation, amount of wash water, and height from which both the feed and the wash water drop to the vanner, density of the pulp, and load per square foot. It will also be important to determine whether the efficiency of the Wilfley table can be increased by adopting a plain eccentric motion in place of the "quick return," which will lessen the violence of the agitation; and produce the travel toward the concentrate discharge end by means of a suitably steep, and adjustable, slope in that direction.

Ore Dressing.

BY ROBERT H. RICHARDS, BOSTON, MASS.

THE group of four papers on ore dressing read at the Butte meeting, all of them dealing with the recent developments at the Great Falls and Washoe plants of the Anaconda Company, form an extremely important contribution to the subject of ore dressing. It proves the efficacy of the new hindered-settling principles first brought out in a paper entitled Close Sizing Before Jigging [1] and later elaborated in a paper entitled Development of Hindered-Settling Apparatus,[2] and it is the first large demonstration on this class of mill that has been made.

It must be very gratifying to all mill men to have these papers at hand, charged with careful investigations and results pointing to the conclusions reached; the papers cannot fail to be a great help to every designer of mills and to mill men who are working on this class of concentration.

The papers have dealt with their subjects so completely and have demonstrated every point from beginning to end in each case so thoroughly it leaves little for the commentator to say in the way of criticism. Therefore it seems best to him to confine himself to adding a few points of history which he believes will be interesting to the readers.

Referring to the paper by Mr. Wiggin, and to the fact that the Richards hindered-settling pulsator classifier was turned down, the writer wishes to say that this classifier was the first of its kind that had ever been introduced, and since that time it has undergone great developments which he believes would overcome the difficulties encountered in the test. In fact, there was one test made at Great Falls which with the Wilfley tables and vanners brought all the tailings down to 0.36 per cent. copper except the fine slime. Since that time, and since the classifier has been further developed, in one Colorado mill it proved a wonderful amalgam catcher, catching in all

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[1] *Trans.*, xxiv., 409 (1894).
[2] *Trans*, xli., 396 (1910).

in a year $30,000, according to the mine report, which was not known to be caught before the classifier was put in, and in another Colorado mill the success was so great that the manager declared if he could not have that classifier he would resign his position.

The writer was honored by Mr. Mathewson by an invitation to walk through the mill with vanning shovel and screen in hand, and by them to call attention to the points at which losses were being made. This was done shortly before the investigations, results of which have now been published, were made. Starting from the $2\frac{1}{2}$-mm. jigs, he examined the tailings of all three sets of jigs, and of the Wilfley tables, and demonstrated that fine free mineral was going into the tailings at every point. He remarked to Mr. Mathewson that if the proper classifier was used this fine free mineral could be so removed from the feed to these machines that it would not appear in the tailings, and instead would appear in the concentrate of machines further down the line; and the writer further stated that he believed the hindered-settling classifier was the machine to do this work.

The special advantages of a good classifier, and also those advantages which hindered settling gains over free settling, are as follows:

1. The feed for jigs and Wilfley tables is so perfectly prepared for those machines that the capacity of the machines can be greatly increased with a classified product over what it would be with a natural product; in fact, Mr. Wheeler has informed the writer that the coarser Wilfley tables in the Great Falls mills have been put up to the capacity of 120 tons in 24 hr. without harming either the concentrate or the tailing. This, of course, could not be done with the finer Wilfleys, but the finer Wilfleys are also increased in capacity. In comparing the hindered-settling classifier with the free-settling, the former is much more efficient in this work of increasing the capacity of the Wilfley tables than is the latter.

2. The fine free mineral, the presence of which in the tailing of jigs and Wilfley tables when fed with natural product was a source of loss, which was pointed out in the paper entitled Development of Hindered-Settling Apparatus, is completely removed by a good classifier, and in this case the hindered-settling does the work better than the free-settling classifier. The fine free mineral which would be in the earlier spigot and therefore in the tailing of the coarser machine, is pushed along to be saved by the later finer machine which is adapted to its treatment. This happens all along the line down to the finest machine, in fact until we have reached the colloidal material, which is so fine that it cannot be treated.

3. The first spigot of a classifier must of necessity be enriched to a very great degree over the later spigots. This was brought out in the paper, Development of Hindered-Settling Apparatus, and the added product which enriches it was called the " added increment," as being those particles of heavy mineral which were there in excess over and above the portion which was a truly classified product. This again is more perfectly done by the hindered-settling classifier than by the free-settling classifier because the ratio between the diameters of the waste minerals and the heavy mineral is much greater in the hindered-settling than it is in the free-settling.

4. The eliminating of slime from the spigots is more perfectly done by the hindered-settling classifier than by the free-settling classifier; in fact, the complete elimination of slime by any free-settling classifier, even the best forms, requires such an increase of water that it is almost impossible and impracticable. The elimination of slime from the spigots by the hindered-settling classifier is practicable and requires no excess of water to accomplish it. When slime is fed with the sand to a Wilfley table it always appears in the head water crossing the tables, and it is a sure sign that the tailings of the Wilfley table are carrying away to waste some fine free mineral that should not be there.

In regard to the papers by Bardwell and Ammon on the hindered-settling principles and the Anaconda classifier, it is very gratifying indeed to see how the first tests described in Close Sizing Before Jigging show that the teeter chamber and the constriction there described have completely realized on a large scale the original predictions.

In connection with this hindered-settling classifier adopted by the Anaconda Copper Mining Co., it is interesting to record that the first hindered-settling classifier which included a teeter chamber above and a constriction below, systematically put together without the use of a pulsating current, was installed in the Wolftung mill, in Boulder county, Colorado, some time about 1907. This classifier was designed by the writer.

Another very interesting point has been brought out by these investigations: namely, the determination of the relative areas of cross-section that are suitable for the constriction and for the teeter chamber. It is evident to a casual observer that there must be some ratio which makes for a suitable teetering condition in the teeter chamber, and at the same time a discharge of the heavier grains down through the constriction, for if the constriction were too small the teetering condition in the teeter chamber might be in perfect

order, but the rising velocity through the constriction would be so great that no sand would go down. This would be the case where the ratio between the two areas was too large. On the other hand, if a straight vertical tube was used, in which the lower part, where the constriction ought to be, was no smaller than the upper part, where the teeter chamber ought to be, then there could be no teeter chamber at all, for the moment that a grain of sand entered the top of that sorting column its fate would be determined; if it was heavy enough to settle it would go down and out without hindrance, and if it was not heavy enough to settle it would go up and out. Hence there must be some ratio which is suitable for each size of grain between the area of the cross-section of the teeter chamber and that of the constriction.

The writer has determined these relative areas by several different methods, but has never reached a final conclusion which was absolutely demonstrated as to just what the ratios between the areas ought to be. He has again and again proved that for coarser sizes the ratio should be smaller, and that for finer sizes it should be larger. The determination of the proper ratios given by Mr. Bardwell is most interesting, and presumably has been demonstrated with sufficient thoroughness to be of permanent value. Besides varying with different sizes of grain, it will probably be found to vary with different specific gravities of mineral, and in consequence for commercial work it will be necessary to adopt some general approximate ratios to do this work. This has been done with the writer's own hindered-settling classifier.

It is also very gratifying to note that the prediction made in The Development of Hindered-Settling Apparatus, that the space between the concentrate and the tailing should be so widened by hindered settling that there would be a valley between the two, has been realized in this work to a very marked degree.

In regard to the paper on the concentration of slime, by Mr. Hayden, the writer thinks a word of history can well be put in here. He had visited a number of the Gates canvas plants in the gold mills of California and had made a special study of them, having in mind the slime problem of the large mills; he also visited in Telluride, Colo., a mill which had adopted the Gates canvas plant, which was successful there in concentrating slime. This Telluride plant, however, did not persist because it was found that cyaniding was more successful for the precious metals, and cyaniding did not work in satisfactorily with the Gates canvas plant.

About this time the Taylor-Woodworth experiment was tried at

Lake Superior on the slime of one of the great copper mills. This process employed a Gates canvas plant with the slight changes necessary to make the discontinuing of the feed and the washing off of the concentrate so nearly automatic that one man could attend 100 or more tables, while by the Gates method one man could tend only 24 tables. For various reason the Taylor-Woodworth plant was not continued.

The writer conceived the idea of making a continuous canvas table in a circular form which should be automatically fed and discharged, and would therefore reproduce the Gates conditions with the labor item almost entirely removed, the table being provided with a rough canvas surface and being run extremely slowly. When he reached this point he discussed it with Mr. Goodale, of the Anaconda Company, and Mr. Goodale immediately tested the idea in the slime plant at Great Falls, which was then in the experimental stage. After trying canvas for a while and finding that the rough surface so obtained encountered difficulties from the greasiness of clay or talc mineral he tried the use of rough cement surface instead, and found that the latter was far better than the canvas, giving no trouble whatever from the greasiness which existed on the rough canvas surface.

The investigators at Great Falls tried many other tests, all of them looking toward simplicity and efficiency, and the resulting slime table now used at Anaconda is that which they developed. It has drifted away from the Gates canvas plant idea suggested by the writer in several points, but it has retained the rough surface and the slow rotation. It is especially gratifying to the writer to find that this round table is winning out so finely and is able apparently to handle the slime problem in these large complex mills quite as well as, if not better than, any of the modern slimers which have been invented. It is particularly gratifying since he has watched the round table losing ground year after year until it has been taken out of nearly all the mills in the United States; the round tables in one of the great copper mills at Lake Superior are the only ones known to him to be in operation at the present time. He has held the ground all along, and believed that he was right, that the day would come when these round tables would come back into the mills and be used on the product for which they were especially adapted. He believes that the tables which have been thrown out by mills have been so thrown out because they were asked to treat a product which was too coarse for them, and as a result their tailing ran too high in values. He believes that if the proper size is fed to the round tables, and a rough surface

is used, the round table will be found to be the best table there is for handling these very fine products.

As Mr. Hayden says, when we get down to the colloidal material—that is, the material which floats in the water like clay—we have got beyond the reach of water gravity concentration.

In winding up these remarks the writer wishes to express his great gratification at the liberal way in which the Anaconda Company has acted in regard to these papers. It is a direct boon to all the engineers, and to the country, that these investigators should have been permitted to put their ideas on paper and give them out to the world. It is to be hoped that other companies will find that their financial and commercial interests are not harmed by allowing a similar work to be made public so that the engineers and mill men of the country can have the advantage of gaining the inside view of the development of ore dressing in all our great mills.

The Evolution of the Round Table for the Treatment of Metalliferous Slimes.

BY THEODORE SIMONS, BUTTE, MONT.

DURING the last half century a great amount of ingenuity and energy has been devoted to the invention of appliances for the recovery of valuable minerals from very fine sands and slimes. The reason for this is that in almost every dressing plant the greatest losses of values, considered relatively as well as absolutely, occur in the treatment of slimes. The natural aim of mill operators to minimize these losses has in recent years received another impetus, from the fact that the gradually diminishing occurrence of high-grade ores makes a more intense recovery of values from existing resources absolutely necessary for profitable operation.

The results of such attempts are found in the appearance of a great number of machines and appliances which make it profitable to-day to rework old dumps, containing the tailings of older processes. A constantly growing difficulty of obtaining skilled labor and the increase in wages made it prohibitory to employ those types of machines which required much attention, and in the middle of the nineteenth century the various concentrating tables of the percussion type, which automatically discharge their products, began to take the place of the intermittent tables, on which, when the bed had reached a certain thickness, operation had to be stopped in order to allow the skimming by hand of the separate layers formed on the table.

In the course of years, two general types of machines evolved for the treatment of fine sands and slimes: The so-called concentrators and vanners, on the one hand, and the buddles and round tables on the other.[1] For the profitable treatment of the very finest slimes, round tables have as a last resort proved more satisfactory and economical than other machines.

In recent years they have again become the subject of close inves-

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[1] Another type of machines, making use of centrifugal force, is at present being tried out at the Washoe works in Anaconda, Mont.

tigation on the part of the mill operators, with a view towards improving the mechanical features of these machines and thus making them surer of success in the competition with shaking tables and vanners, which during the last decade have to a large extent usurped the function of slime-treatment machines.

It is the intention of this paper to outline the evolution of the so-called round tables from the form in which they first appeared in the dressing plants of Germany and Austria to the various forms in which they are found to-day in the dressing plants the world over.

GENERAL PRINCIPLES.

The ultimate success of a machine depends in the first instance on its strict compliance with the laws and principles controlling the work to be performed. No amount of mechanical perfection can assure success when these laws and principles are violated.

To enable the reader to draw correct conclusions as to how far the tables described in this paper have complied with fundamental principles, and in order to point out in which direction future improvements must be looked for, the writer deemed it helpful to explain briefly the principles and fundamental laws which underlie a separation of minerals on so-called round tables as well as on other machines. They are generally known under the name of film sizing.

FILM SIZING.

If we spread out, in a very thin film, on a slightly inclined plane, water-sized (classified) pulp containing equal-falling particles of mineral, the water running down the incline will not attack all the points of each particle with the same force as in a deeper stream, but the larger particles of lesser density will be exposed at their higher points to a greater force than the smaller, dense particles, because close to the deck of the table the water, due to adhesion, has a lesser velocity than in the upper part of the film.

The result is, that, given a mean velocity of the water, which depends on the inclination of the table, the coarser particles of less density will be carried away by its force, while the denser—that is, the finer—particles will resist the action of the water and remain clinging to the tables. In this manner a separation is made, in which the specific gravity (density) of the particles seems to play the chief rôle.

An important requirement for a separation of the equal-falling particles on an inclined plane is a thin film of pulp and water, because only in a thin film can a difference in the force of attack on the

individual points of the particles of different size take place.   If the water flows over the incline in a deep stream all points of each particle are attacked by the water with nearly the same force, and a sorting action will take place according to the law of equal falling.

Therefore, to subject a classified pulp to the action of a deep stream of water is practically the same as subjecting it to another process of classification, which may turn out to be more perfect than the preceding one; but will not effect a separation or concentration of the values.

A further important condition for a successful separation is the proper velocity of the flow of pulp and water over the table; which depends likewise on the inclination of the table.   Up to a certain inclination the stream of water flowing over the layer of pulp will have no effect on the particles, because the velocity of the water is too small to move them.   If, on the other hand, the inclination is too steep, the velocity will be so great that the denser particles will be carried away with the less dense ones.   Only a mean inclination of the table, and therefore a mean velocity of the pulp and water, will permit a separation according to specific gravity.

Another requirement for successful separation is the proper consistency of the pulp.   If too thick, that is, if the percentage of solids in the diluting water is too great, the water cannot act undisturbed upon the individual particles contained in the pulp because of interference of the particles among themselves.   If the pulp is too thin, larger table area is required to do the same amount of work; in other words, the capacity is unnecessarily diminished.

What the consistency of the pulp should be in any individual case is a problem that can only be satisfactorily solved by careful experiments and trials.

Film Sizing Appliances in General.

Appliances that depend partly on separation by falling through a horizontal current of water and partly upon the differential rate of travel along the bottom of the stream have been in use from immemorial periods. They constitute the most primitive type of a dressing plant and illustrations of them are found in Agricola's *De Re Metallica*, published iu 1556 (see Figs. 1 and 2).   They are found in almost identical, but doubtless independently invented, forms among semi-civilized races all over the world.

Buddles.

As usually constructed, these consist of a rectangular inclosed wooden trough, at the head of which the pulp to be treated is fed in,

an additional stream of water being sometimes run in near the head of the trough. The inclination of the trough, the quantity of water, and the resulting velocity of the current are so proportioned to the size and weight of the particles of mineral that the heavier mineral remains at rest in the trough, while the lighter is carried off. In order to prevent particles of the lighter material from being entangled in and held back by the heavier, it is necessary to work the material lying on the floor of the trough up against the current, which is usually done by means of a broom, rake, or hoe. The clean
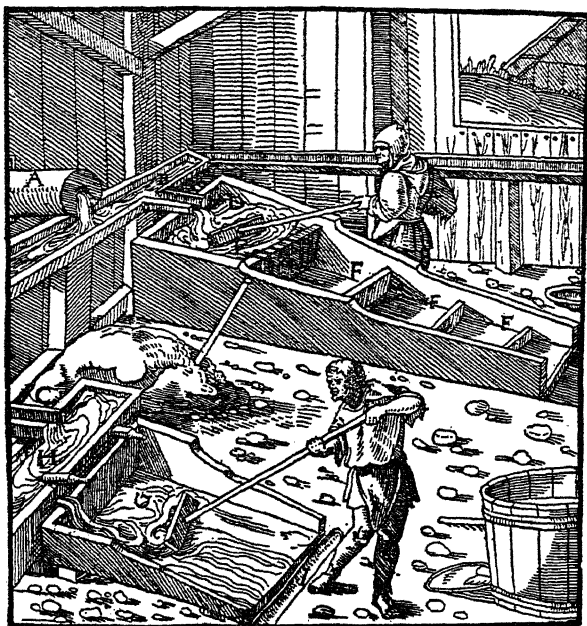
A—Pipe.   B—Cross launder.   C—Small troughs.   D—Head of the Buddle.
E—Wooden scrubber.   F—Dividing boards.   G—Short strake.

Fig. 1.—From Agricola's *De Re Metallica*, Published in 1556.

heads thus obtained are usually shoveled or raked out from time to time, while the water runs off continually. Fig. 3 shows a buddle as described above.

Round Tables in General.

From the rectangular buddle described above, developed in the natural course of evolution the round table.

A large number of very narrow rectangular buddles arrangéd around a common center, with their longitudinal dividing walls removed and the vacant sectors between the buddles filled out, form

the so-called round table, either of the form of a cone (convex) or that of a funnel (concave), according to whether the inclination of the surface is away from or towards the center.

The principle of action of the round table is identical with that of the rectangular buddle, except for the fact that the velocity of the pulp as it flows down the table is not uniform, because on a convex table the pulp spreads out over a larger surface, while on a concave surface it is crowded into a smaller area. In the first case the thickness of the film of pulp and therefore its velocity gradually decrease,
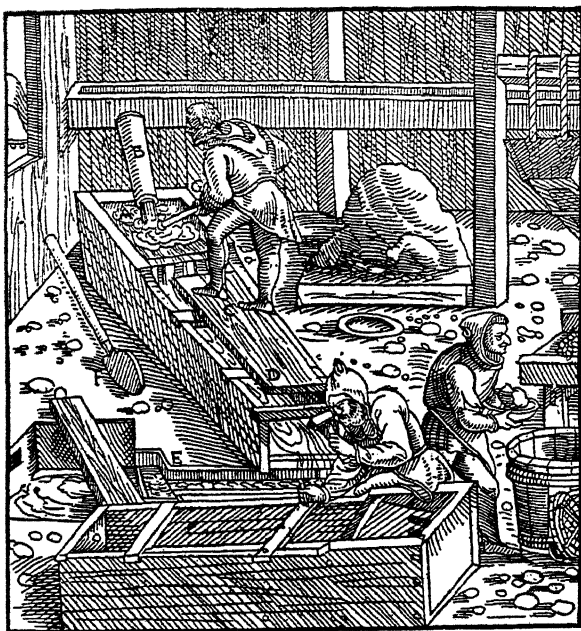
A—Head of buddle. B—Pipe. C—Buddle. D—Board.
E—Transverse buddle. F—Shovel. G—Scrubber.

Fig. 2.—From Agricola's *De Re Metallica* (A. D. 1556).

while the reverse takes place in the concave or funnel-shaped table. The less dense particles rushing towards the discharge end of the table will therefore accumulate at this portion of the table in a thicker mass in the case of a convex table and in a thinner mass in the case of a funnel-shaped table. Inasmuch as these particles are destined to go to the tailings dump or to another machine, this feature does not seriously affect the concentration of values, which takes place in the upper portion of the table.

Round Tables of the Intermittent Type.

The first round tables, like the rectangular buddles, were intermittent machines; that is, after having been fed with pulp and water for a certain period, the operation had to be stopped to allow the skimming of the various layers of material formed.

Convex Round Tables—Intermittent Action.

Figs. 4 and 5 illustrate one of the old constructions of a convex round table. The table *A* forms a ring with an outside diameter of 20 ft. and an inside diameter of 6 ft., which gives a radial length of the table of 7 ft. The deck consists of boards, nailed on the sills *B,* which are arranged star-shaped. The surface is carefully trued with a plane. The lower end of the deck is surrounded by a board wall,
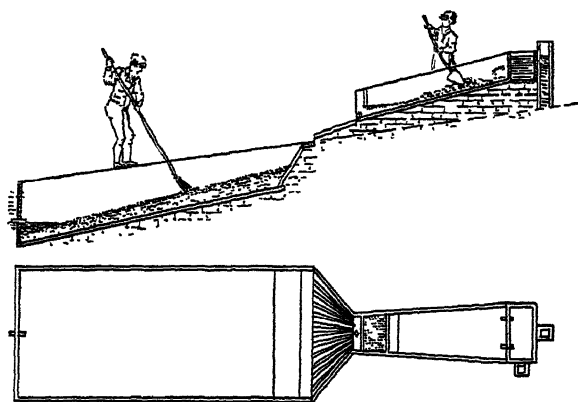
Fig. 3.—Rectangular Buddie.

A_1, from 9 to 12 in. high. The openings *O* in this wall permit the discharge of the tailings into the tailings launder *R*. The inner concentric wall A_2, which is 12 in. high, is joined by the conical distributing apron *H*, which receives the feed through the funnel *C*. The center post *K* carries the step bearing for the spindle *S*, which at its upper end is supported by the bearing L_1 and is revolved by means of the gearing *T* and the pulleys *P*.

The pulp is brought to the table in the launder R_1, which delivers it into the funnel *C*, whence it flows on the table via the apron *H*. Two sockets C_1, cast integral with the funnel, carry the arms *D* on which are fastened the windlasses *N* and N_1 by which the brushes *F* and F_1 can be adjusted. These arms with their brushes make from 10 to 12 rev. per min. The object of the brushes is to keep the layer of ore

smooth. In proportion as it increases in thickness the brushes are raised by means of the windlasses.

One drawback of these tables is, that, due to the variable current of the pulp, the bed forming upon the deck changes its inclination and that it remains too loose, permitting the formation of furrows,
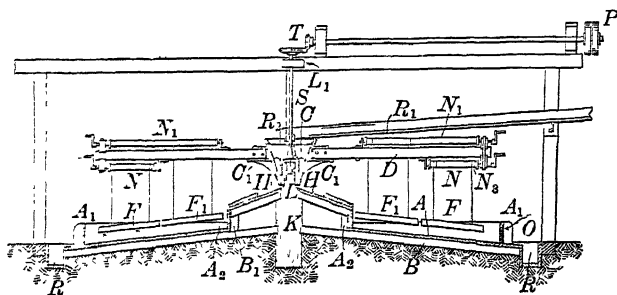
FIG. 4.
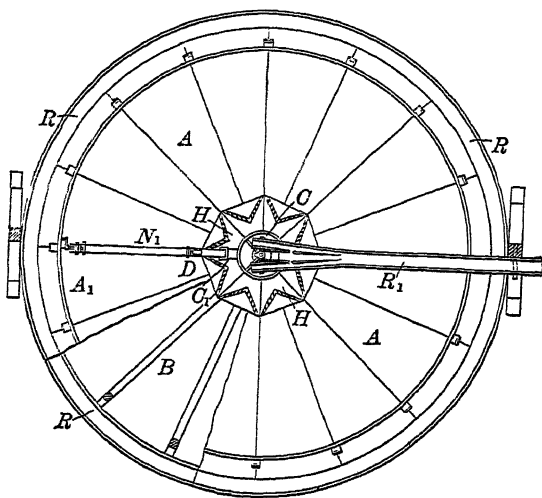
FIG. 5.

FIGS. 4 AND 5.—CONVEX ROUND TABLE.

which prevent a perfect separation of the minerals. It is therefore not possible to make a clean product in one operation, but the several products must be re-treated on another set of tables.

The work is intermittent, due to the necessary stops for skimming. It takes from 2 to 3 hr. for the table to form a layer ready for skimming.

Convex round tables of the intermittent type similar in design and operation to the one shown in Figs. 4 and 5, but making use of masonry instead of wood construction, are also built. (See Figs. 6 and 7.)

Strips of canvas
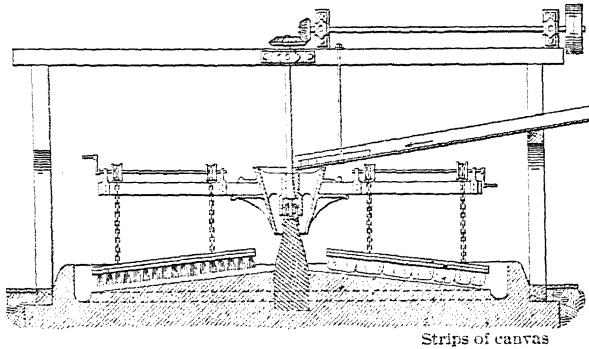
FIG. 6.
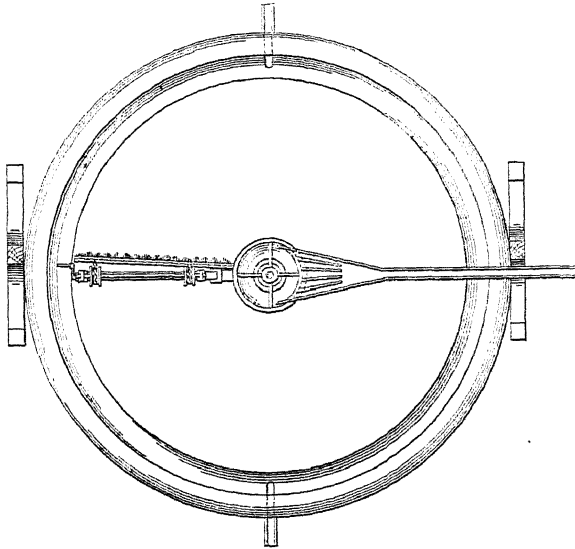
FIG. 7.
FIGS. 6 AND 7.—CONVEX ROUND TABLE, MASONRY CONSTRUCTION.

CONCAVE ROUND TABLE—INTERMITTENT TYPE.

The substructure of the concave or funnel-shaped table is very similar to that of the convex table, as will be seen from the illustrations, Figs. 8 and 9.   The deck A, which rests on the sills B, is surrounded by a wooden wall A_1 from 12 to 15 in. high, supporting the conical feed apron H which receives the pulp for distribution on the table.

In the center the table is cut off by a wooden barrel A_2, 6 ft. in diameter and extending 12 in. above and 18 in. below the deck sur-face. The upper part of the cylinder is perforated to allow the pas-sage of the tailings, which are carried off by the tailings launder R.

The distribution of the pulp, which is brought to the table by the launder R_1, is effected via the funnel C, from which radiate the four
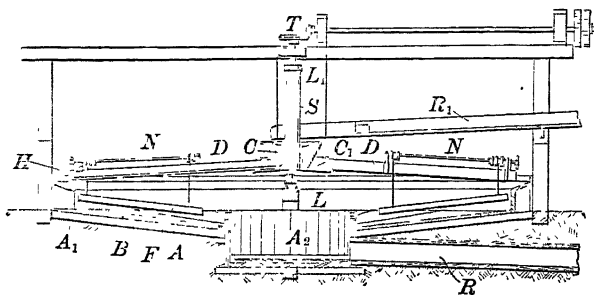
FIG. 8.
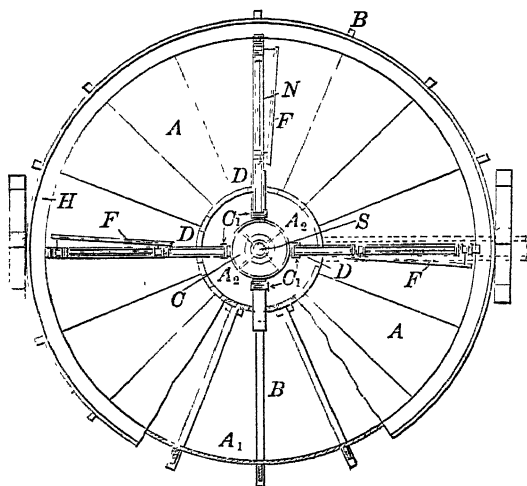
FIG. 9.

FIGS. 8 AND 9.—CONCAVE ROUND TABLE.

launders D which spread the pulp on the feed apron H. The laun-ders D also carry the windlasses N by means of which the position of the brushes is regulated.

The spindle revolves 10 times per minute. Owing to the fact that the four revolving launders distribute the pulp more uniformly over the table, the surface of the bed of minerals remains smoother on a con-cave than on a convex table, which results in better work.

The principles of operation of the concave table are practically the same as those of the convex tables. It is likewise intermittent.

The great drawbacks of intermittent action were soon realized, and efforts were directed to the construction of tables which discharged the various products automatically, thereby making the operation a continous one.

ROUND TABLES OF THE CONTINUOUS TYPE.

The principle of continuous action was first applied to the round table by making the table itself rotate, and allowing the pulp as it travels down the incline to come under the influence of stationary sprays of water, which wash the various products from those portions of the table on which they are formed, and as soon as they are formed, into suitable collecting launders.

In this manner a clean surface is ready to receive another layer of pulp when the table has made a complete revolution; in other words, the action is continuous.

Having thus achieved continuous action, capacity was the next item of importance. So, rotating tables were made of larger and larger diameter. In this process of evolution two obstacles were encountered. As the tables grew larger, they required more extensive and costly foundations and substructures, occupying excessive space in the mill. Then they became so heavy as to require an excessive amount of power, and, being unwieldy, the motion became less steady, and as a consequence the separation less perfect.

These obstacles were overcome: the first, by placing several decks on the same central shaft one above the other; the second, by making the table stationary and allowing the fixtures, consisting of the feed launder, the spray pipes, and the collecting launders, to revolve.

All modern round tables belong to one or the other of these types as they evolved from the first rotary round table, installed in the mills at Clausthal in the Harz mountains in 1853.

In principle, continuous-acting round tables have not undergone material changes since their introduction. But numerous mechanical improvements have been made, chiefly in the choice of material for the deck covering. Wood, which was used in the construction of the first tables, was subject to the undesirable property of warping. It was first replaced by cast iron, which for a great number of years was used for deck material as well as for other parts of the machinery, although the turning of the castings to a true surface was exceedingly expensive then.

In the beginning of the twentieth century a covering of rubber

was tried on a wooden substructure. The first experiments with this material were carried on in the Harz mountains at the Hülfe Gottes mine. The technical results were altogether satisfactory, but the cost of this material proved too high in the long run. Next a cement cover was tried with a thickness of 3 cm. (1.25 in.). It was found that this cement cover would crack through the cold of the winter. To prevent this a series of nails was driven into the deck before the cement cover was applied, these nails being set 10 cm. (4 in.) apart and protruding slightly above the deck.

This precaution, however, did not altogether prevent cracking, and new experiments with covering material were made, with the result that a concrete foundation with a thin cement surfacing proved in the end to be the cheapest and most satisfactory deck covering. A 3-cm. (1.25-in.) layer of concrete was placed on the deck, made of rough boards, and finished up with cement. This also permitted the surface to be made more or less rough, depending on the character of the slimes to be treated.

A concrete cover was first employed on the so-called Harz two-deck round table (Fig. 10), the upper deck being funnel shaped, the lower cone shaped, both decks being fastened to and revolved by the same main shaft.

This arrangement of decks had the great advantage of making possible the finishing of the product on the lower deck without taking up much mill space, although at the expense of much water. On the other hand, the increased weight of the table due to the concrete and cement covering made it impracticable to lubricate properly the toe and step bearing of the main shaft. In the end it became necessary to separate the decks, and at first the upper deck was made revolving while the lower one remained stationary.

This new method of covering the table with concrete and cement had the other great advantage that it permitted giving the table *in loco* an inclination best adapted to the nature and the consistency of the slimes as found by trial. It also permitted a ready change of this inclination should a change in the nature of the feed demand it.

One disadvantage of these tables is, that a small unevenness of the surface, which even with the greatest skill in turning will occur, is apt to cause an accumulation of the heavier material in concentric rings, particularly when the table is overfed.

In recent years iron decks have come into use again, for various reasons. First, the technical improvements in the manufacture of iron appliances have made the manufacture of iron decks much cheaper than in previous years. An iron deck is also much lighter

than a wooden table with a concrete cover, and hence requires less power. In round tables of the percussion type, like the Bartsch table, wooden decks would not offer the necessary resistance, so that in these machines iron decks are used.

EXAMPLES OF ROUND TABLES OF THE CONTINUOUS TYPE.

1. *Revolving Tables with Stationary Fixtures.*

Figs. 11 and 12 show a revolving round table with stationary distributing and receiving launders, and stationary wash-water pipes. The deck *A* has the shape of a slightly inclined cone and, if made of wood, it rests on a cast-iron plate which is supported in the center by a spider *B*. The latter is keyed to the vertical spindle *D*, which at the bottom revolves in a step bearing *E*, being driven at the top by
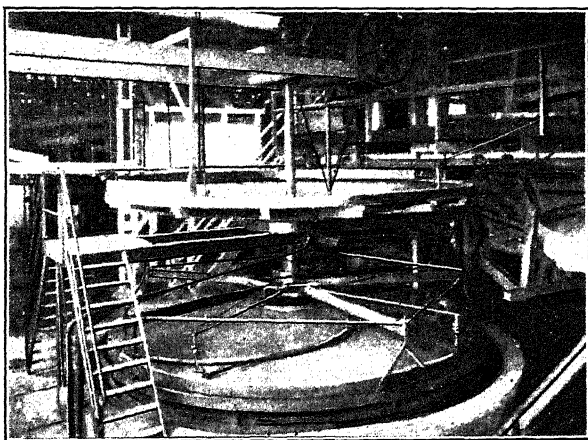
FIG. 10.—HARZ DOUBLE-DECK REVOLVING TABLE.

a worm wheel *F*, through a worm *G*. The latter forms the end of a horizontal shaft provided with a driving pulley *H*.

The distributing launder *J* is a ring-shaped vessel divided into two compartments. The smaller one receives the pulp through the feed launder *R*. Into the larger one flows the wash water from the circular tube *C*, which also provides the spray pipes C_1. From the distributing launder *J* pulp and water reach the tables through small short pipes in the bottom of the launder.

As the table revolves, each part of it passes under the pulp compartment and immediately after under the wash-water compartment of the distributing launder *J*, remaining under the influence of the latter until it again passes under the pulp compartment.

The operation is continuous.   The coarsest, barren mineral particles are immediately washed off the table into their respective section of the receiving launder L.   A middlings product and the concentrates which settle on the upper portion of the deck are washed off by the spray water into other sections of the receiving launder.
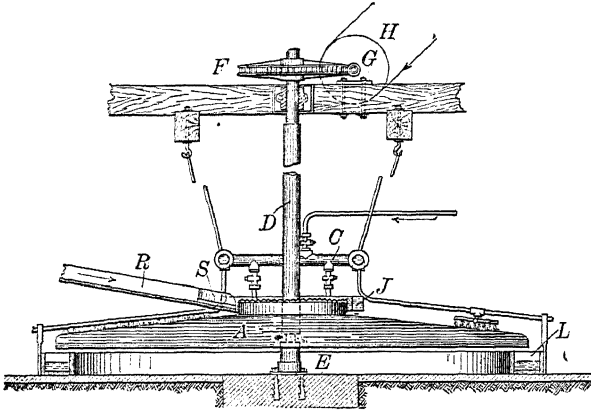
FIG. 11.
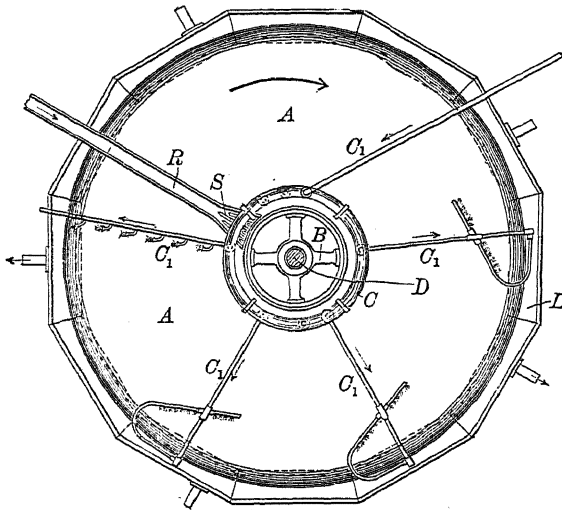
FIG. 12.

FIGS. 11 AND 12.—REVOLVING TABLE WITH STATIONARY FIXTURES.

The diameter of the table varies with the fineness of the slimes to be treated.   A common dimension is from 16 to 24 ft.   The inclination varies from 1 to 10, to 1 to 12.   The table makes from 15 to 20 rev. per hour, corresponding to one revolution in 3 to 4 min.

The average capacity is 10 gal. per minute of slimes containing from 8 to 10 per cent. of solids. The quantity of wash water is 25 gal. per minute. The power required is 0.25 h-p.

In a more modern construction of revolving table, Fig. 13, the substructure consists of iron beams and corrugated iron, which is followed by a cement cover, forming the deck.

The Evans Slime Table.—In the early seventies there appeared in the copper mills of the Lake Superior district revolving round tables patented by Mr. Evans, which differed from the tables theretofore in use by having a stationary conical feed apron or head, which extended half way down the incline and was supposed to protect the headings formed on the revolving portion of the deck until they were ready to be washed off.
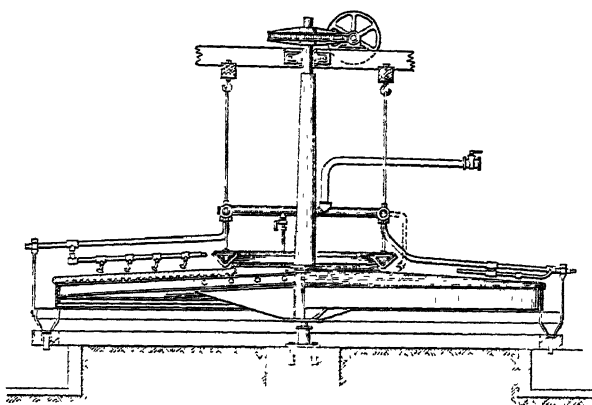
FIG. 13.—REVOLVING TABLE WITH IRON SUBSTRUCTURE.

Fig. 14 gives the main outlines of the table. *A* is a launder to conduct the slimes from the catch pit or slime box to the distributor *B*, which has a partition B_1 to separate the clear water from the puddled water or slime water. The clear water is supplied by pipe *P* to the distributor, and runs over one-half of the table, while the slime water runs over the other half, being controlled by the division piece *L*. The sand and water being on one side of distributor *B* run through its perforated bottom, and are distributed equally over one-half of the stationary head *C*, and run on the rotating table *D* into the circular launder *N*, then through the waste pipes *O O*.

The headings remain on the upper part of table *D*, and after concentration are shielded from the action of clear water by passing under the spiral-shaped stationary head *C*, until they reappear at the end of the revolution from under the widest part of the apron. Through the action of clear water the proper grades of ore are

washed about half-way down the rotating table *D*. They then come in contact with the diagonal perforated pipe *E*, and are rewashed by a succession of small jets from perforations of small pipes. The ore passing between the jets is carried around the rotating table *D* until it comes in contact with a jet of water from pipe *F* and conducting board *G*. The jet *F* conducts the ore into hutch *H* through pipe *I*.

The middle or second-grade ore is washed off table *D* by the perforated pipe *E*, and is deposited in hutch *J* through pipe *K* to be rewashed.

The speed of the machine is one revolution in 80 sec. The pitch or incline of the table is 1.25 in. to 1 ft. The pitch of head is 1.75 in. to 1 ft. The capacity of the machine is from 25 to 35 tons per day of 24 hr.

The same type of table was afterwards used at the Washoe works in Anaconda, Mont., for treating copper slimes. To save mill room
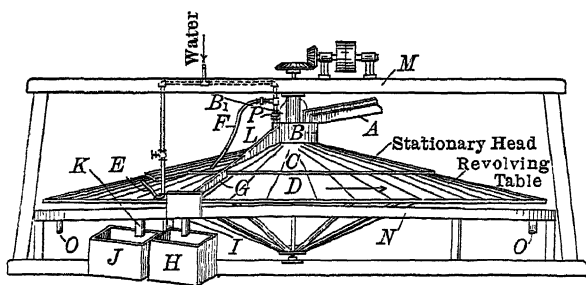
FIG. 14.—EVANS SLIME TABLE.

the tables were built with two decks on the same shaft, as shown in Fig. 15.

At these works the construction and operation in the near future of a 20-deck revolving round table is under contemplation. This subject, I am informed, is treated in a separate paper to be presented at this meeting.

2. *Stationary Tables with Revolving Fixtures*.

· *The Linkenbach Round Table*.—In 1878 Linkenbach designed for the Ems Lead and Silver Works, in Germany, a round table which is the prototype of all tables belonging in this class.

Figs. 16 and 17 illustrate an early construction of this table. It consists of a conical deck *A* with a cement surface on a rough masonry foundation. Before hardening, the deck is turned true and smoothed. Where mill space is abundant this deck can easily be

made of larger diameter, this being one of the chief advantages of this type compared with the revolving table.

The vertical spindle A_1 carrying the fixtures revolves at the bottom in a step bearing M, while to the upper end is keyed the worm wheel N, which is set in motion by the worm E. The latter forms the end of the horizontal shaft F provided with a driving pulley H.

The feed is delivered through pipe I into the revolving distributing launder J, which is suspended from the pipes K. Below it, likewise suspended from K, is the wash-water launder L. The water is distributed throughout the fixtures by way of the hollow spindle A_1.

Suspended from the revolving beams B and revolving with them is the collecting launder G, the individual sections of which deliver their respective products into the stationary sump launders O, O_1, O_2, O_3, by means of pipes of different lengths.
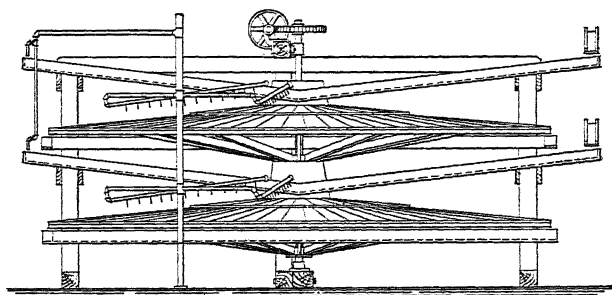
FIG. 15.—OLD STYLE ANACONDA DOUBLE-DECK EVANS TABLE.

The separation and automatic discharge of the minerals take place in the same manner as in the rotating tables, with this difference : That the change of the place of delivery where the pulp is spread on the deck is caused by the revolution of the fixtures in the first case and by the revolution of the table itself in the second case.

The wash-water pipes and spray pipes are so arranged that their positions can be changed within certain limits. This permits the pointing of the streams of water in the direction which they must assume in order to wash the various products formed on the deck into the respective sections of the receiving launder.

To save mill space, Linkenbach tables were built with three decks, one above the other. Fig. 18 shows an installation of tables of the Linkenbach type built by Fried. Krupp in Germany.

They are built with diameters of from 19.5 to 32.75 ft. The capacity of the largest size is from 2,000 to 2,400 lb. of material per hour. The number of revolutions is from 0.25 to 0.43 per min. The power

required is 0.75 h.-p. The clear water consumed per minute is from 48 to 55 gal.

Principle of Operation of the Linkenbach Table.—The action of this table is continuous; that is, the pulp is spread on the table and is al-
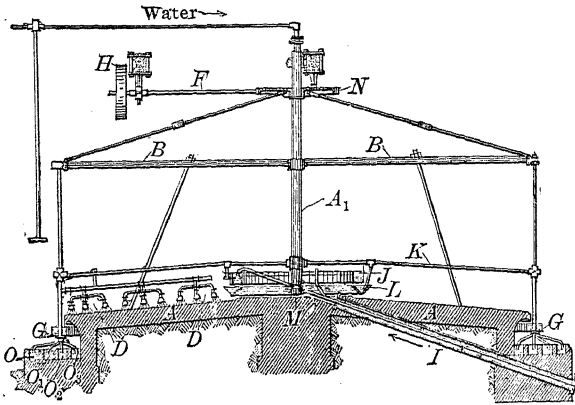
FIG. 16.
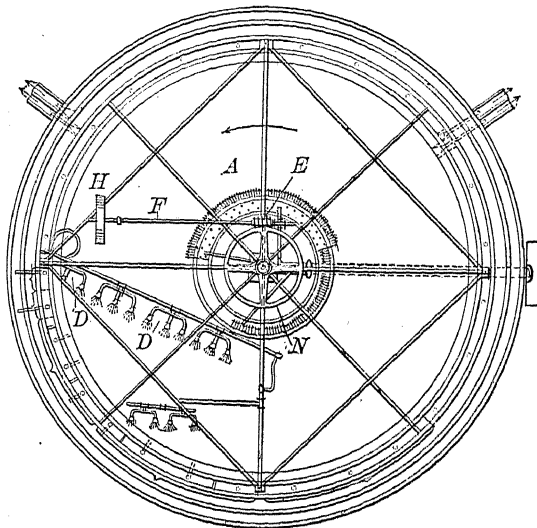
FIG. 17.

FIGS. 16 AND 17.—EARLY FORM OF LINKENBACH ROUND TABLE.

lowed to separate into various layers, which are washed off successively into respective sections of the receiving launder, which revolves with the fixtures.

If we observed any particular section of the table during a complete cycle of operation we would see, if conditions were ideal, approximately the following picture: In flowing down the table the individual particles of the pulp will settle on the deck in concentric rings according to their specific gravity, the densest nearest to the top and the less dense nearest the circumference. If we have a pulp containing, for instance, galena and blende as the valuable constituents with, say, quartz as a gangue, that section of the table over which the pulp distributor has just passed will immediately be exposed to the action of the wash water, which washes the gangue off the table into its respective section of the collecting launder. There remains then on that section of the table near its circumference a ring
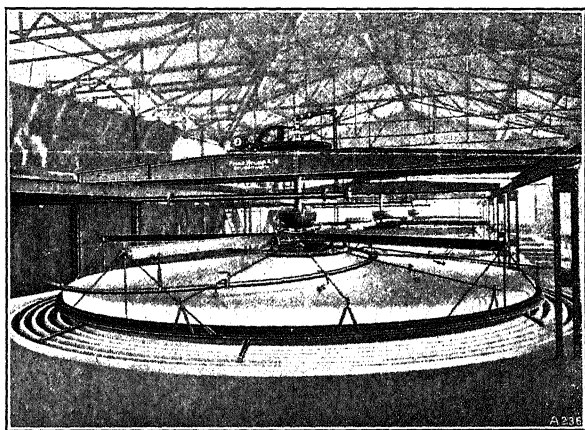
FIG. 18.—TABLES OF THE LINKENBACH TYPE, BUILT BY FRIED. KRUPP, GRUSONWERK, GERMANY.

of the next dense particles, consisting of middlings containing blende, some galena, and some quartz. This layer of material is now attacked by the first set of spray pipes that come along in the course of revolution and is washed into its section of the receiving launder. Next the layer of blende higher up is attacked by a following spray, then a layer of blende mixed with some galena still higher up, and at last the pure galena particles nearest to the top of the table, all being washed into separate compartments of the receiving launder. After the last spray has passed that section of the table, it is completely cleared to receive a new layer of pulp from the pulp distributing launder, which follows in the wake of the last water spray.

By the proper distribution and direction of the sprays in relation to the pulp distributor a more or less perfect separation can be made, as well as a number of finished products, middlings, and clear tailings.

3. *Percussion Round Tables.*

The success of percussion tables of the rectangular type for sepa-
rating valuable minerals from their gangue, particularly when sev-
eral metals are to be recovered, as, for instance, from an ore containing
galena and blende, has led to the application of the percussion prin-
ciple to round tables.
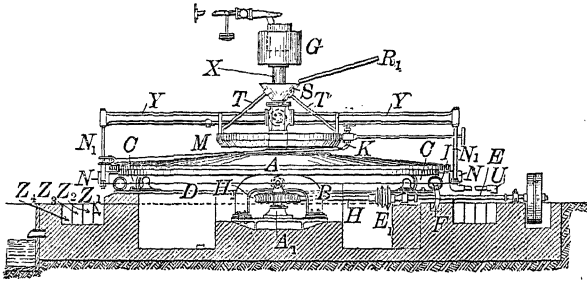
Fig. 19.
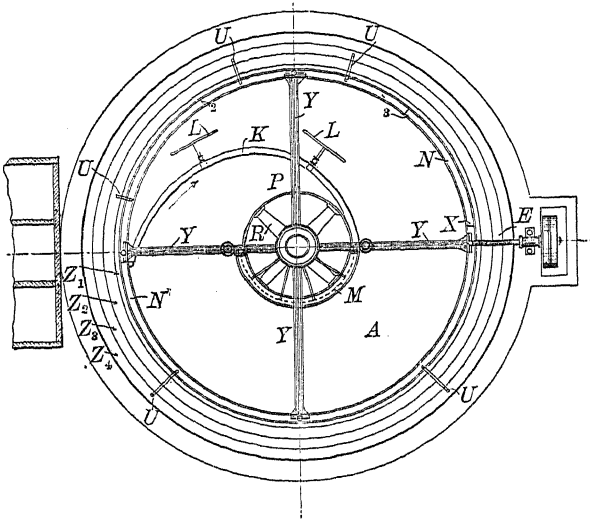
Fig. 20.
Figs. 19 and 20.—Bartsch Round Table.

A well-known table belonging to this class is the Bartsch table,
which is operating successfully in various dressing plants in Ger-
many.   It is manufactured by the Humboldt Engineering Works Co.
at Kalk, Germany.

The Bartsch Round Table.—As illustrated in Figs. 19 to 23, this table
consists of a cone-shaped deck *A* made either of two cast-iron plates,

turned true and covered with a coat of durable paint, or of a rubber sheet stretched over a wooden substructure. The spindle X, supported by a step bearing A_1, moves freely within the spider R, the latter forming the central support of the table, which on its circumference rests on the rollers C. To the central part of the spindle are
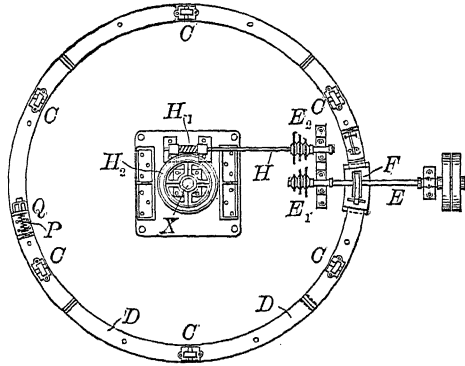
FIG. 21.
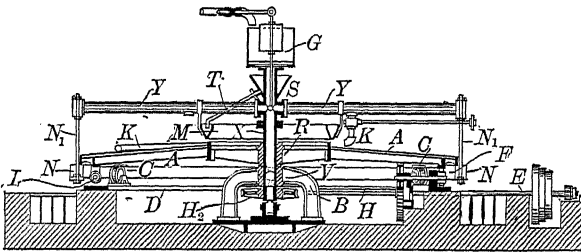
FIG. 22.
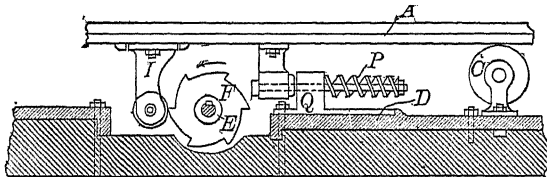
FIG. 23.

FIGS. 21 TO 23.—THE BARTSCH ROUND TABLE.

fastened the four hollow arms Y on which are suspended the distributing launder M, the curved spray pipe K, and the rods N_1 which carry the collecting launder N.

As will be seen, the curved spray pipe K with its auxiliary sprays L starts from the end of the distributing launder M. All sprays

receive the necessary wash water under constant pressure through the hollow arms Y from the water tank G. The latter is provided with a float regulating the inlet valve. The spindle X also carries the funnel S which receives the pulp through the launder R_1. From the funnel it flows through the pipes T into the distributing launder M, which feeds the table.

The mechanical operation of the table is performed by the spindle X, which not only rotates the fixtures, consisting of the distributing launder M, the spray pipes K and L, the funnel S, and the collecting launder N, but also imparts to the table the tangential shocks in the following manner:

On the shaft E is keyed a cam F, which strikes a roller, carried on a bearing I underneath the table, at the moment when the table, due to the tension of the spring P, is in its artificial state of rest against the stop Q.

During the forward thrust the spring is further compressed, and at the end of the motion the table is jerked back against the stop Q, the direction of the jerk being opposite to that of the rotation of the fittings. By means of sprocket wheels, and chains E_1, E_2, the motion of shaft E is transmitted to shaft H, at the end of which is a worm H_1 which rotates the worm wheel H_2 at the bottom of the spindle X. This arrangement permits of a change in the speed of revolution whenever desired.

Fig. 21 shows a somewhat different arrangement for regulating the speed of revolution by the use of step pulleys in place of sprocket wheels, and the shaft F which carries the cam is set in motion by a spur wheel on shaft E.

The pulp is spread on the table through the perforations of the distributing launder M, and if the table has the proper inclination the gangue will leave the table, while the valuable (denser) minerals will cling to it until they come under the influence of the spray water.

The individual products are washed into the respective sections of the collecting launder N, which rotates with the other fixtures, and they are finally discharged through the pipes U, of different length, into the stationary circular launders Z_1, Z_2, Z_3, Z_4.

The diameter of the table is 13 ft. It is capable of treating about 0.5 ton of slime per hour. The forward thrust of the table producing the shock is from 10 to 20 mm. During one-half revolution from 100 to 120 shocks are imparted to the table. A complete revolution of the fixtures takes place in about 2 min.

From 0.25 to 0.5 h-p. is required for operation. The consumption

of spray water per hour is from 12 to 18 gal. for light material, and up to 35 gal. for heavy material.

Principles of Separation of the Bartsch Percussion Round Table.— While gliding down the incline, the tangential shocks imparted to the table compel the particles of mineral to follow a path which is the resultant of the straight path down the incline and the path at right angles to it, due to the shock.

The distance traveled in the direction of the incline is greater, the less dense and the larger the particles, while the distances traveled in the direction of the tangential force causing the shock are practically the same for all particles. This is due to the fact that the weight and therefore the momentum of these particles is approximately the same.

As a consequence, the resultant paths traveled by the individual particles will form different angles with that generatrix of the cone on which the said particles started on their journey down the table, the magnitude of the angle being a function of density, the largest angles corresponding to the densest, the smallest angles to the least dense particles.

This process is repeated during each time interval between two successive shocks, and each individual particle is advanced not only into another generatrix of the cone, but also into a larger parallel circle of the table; that is, towards the lower circumference of the same.

The distances between two parallel circles measured in the direction of the generatrix, due to acceleration, become greater and greater, while the angle formed between the resultant path of the particles and the generatrix in which this particle happened to be at the beginning of the last time interval, becomes smaller and smaller.

If we connect the positions occupied in succession by one and the same particle during its travel over the whole table surface, we obtain a curve, which in its horizontal projection corresponds to a spiral.

Since the individual particles of a classified feed have different densities, we shall get different curves which will be more or less steep. The least steep curve which intersects the circumference of the table at a point farthest from the generatrix on which the particle started its journey, corresponds to the densest particle, and *vice versa.*

If we treat, for instance, a pulp containing as valuable minerals galena and blende, with a gangue of, say, quartz and graywacke, the table, after being set in operation, will present at any individual moment the picture shown in Fig. 24.

The pulp is spread upon the surface 1–2–3–4, and the gangue is

washed into the section E_1-E_2 of the collecting launder N. On the surface 2–3–5–6 the separation of a second-grade (less dense) blende product takes place, which is washed into the section E_2-E_3 of the collecting launder; on surface 5–6–7–8 is made a first-class blende product and washed into section E_3-E_4 of the launder; finally a second-class galena product is separated out on surface 7–8–9, and a first-class galena product on surface 8–9–11–10–8, and washed respectively in sections E_4-E_5 and E_5-E_1 of the collecting launder.
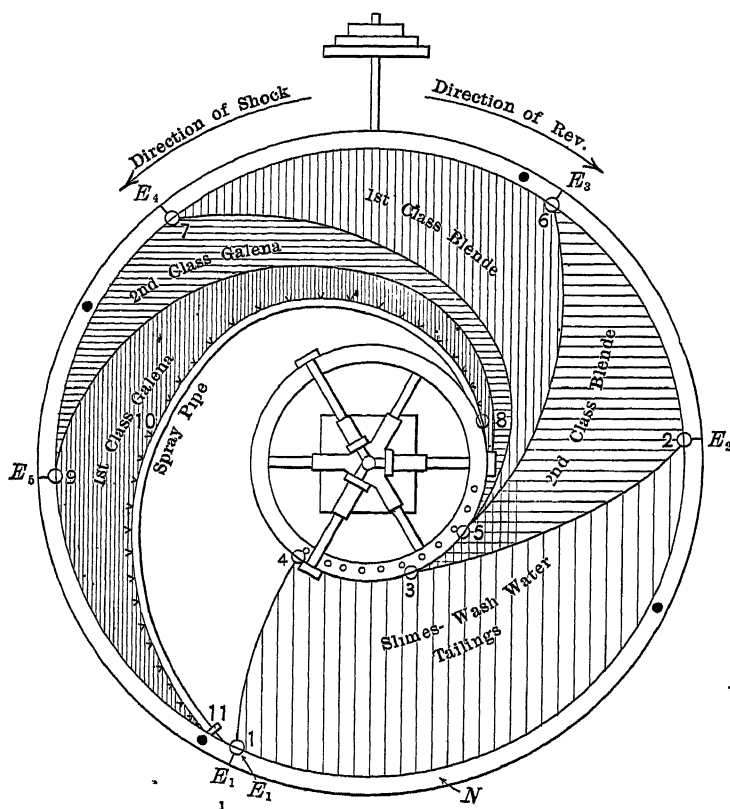
FIG. 24.—DISTRIBUTION OF PRODUCTS ON THE BARTSCH ROUND TABLE.

4. *Revolving Round Tables with Adjustable Inclination of the Deck (System Demuth).*

Various attempts to construct tables with adjustable slope have been made, with more or less success.

Such a table is built by the Firma Groeppel in Bochum, Germany, and is illustrated in Fig 25. At this moment the writer has at his disposal no drawing showing the details of construction of this table.

It appears, however, that the sections which compose the table over-lap each other slightly, and are fastened at their upper ends to a sleeve which can be moved up and down the main vertical shaft, thereby changing the slope of the deck within certain limits. The deck covering, which is either rubber or linoleum, is in one piece.

Round Tables with Conoidal-Shaped Decks.—The generatrix of the deck surface of all tables described in the preceding pages is a straight line. At the Reduction Works of the Boston & Montana Mining Co. at Great Falls, Mont., tests are being made at present with round
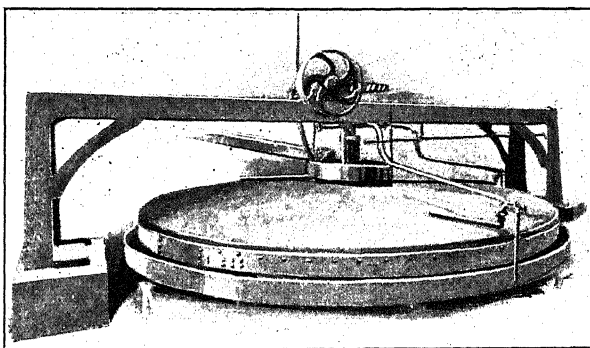
FIG. 25.—REVOLVING TABLE WITH ADJUSTABLE SLOPE. SYSTEM DEMUTH.

tables, having a deck the generatrix of which is the arc of a circle, whose chord has an inclination from the horizon varying with the fineness of the pulp to be treated.

The description of these tables and the results obtained, I am told, form the subject of a separate paper to be presented at this meeting.

It is the writer's belief that by referring to these tables he has arrived at the latest stage in the evolution of round tables, and there-fore at the end of the task which he started out to accomplish in presenting this paper.

Roasting and Leaching Tailings at Anaconda, Mont.

BY FREDERICK LAIST, ANACONDA, MONT.

WHILE remodeling No. 1 section of the concentrator at the Washoe Reduction Works of the Anaconda Copper Mining Co. during the summer of 1912, for the purpose of ascertaining what additional metal recoveries could be made by improvements in concentration practice which had been developed by the staff of the Boston & Montana Reduction Department, at Great Falls, it was deemed advisable to carry on some experiments on the treatment of the regular mill tailings by roasting and leaching, to see whether the losses of metal values might not be still further decreased.

In the course of these experiments about 5,000 tons of tailings were roasted, and of the resulting calcine about 200 tons were leached. The results were so satisfactory that it was decided to continue the work on a larger scale, particularly as regards the leaching end of the process, and construction work on an 80-ton roasting and leaching plant was accordingly commenced during the latter part of February, 1913, and is being pushed as rapidly as possible.

80-Ton Leaching Plant.

This plant will consist of one 20-ft. roasting furnace of the MacDougall type, specially adapted to the work of roasting for leaching; two leaching tanks, 32 ft. in diameter by 12 ft. deep, together with the necessary solution tanks, air lifts, precipitating launders, etc.

It has been decided to precipitate the copper from solution on scrap iron, so as not to hamper the roasting and leaching departments by any weakness that might develop in the precipitating department. It is, however, the intention to experiment with a number of different methods of precipitation; for example, by electrolysis and by hydrogen sulphide.

The acid required for the operation of this plant will be purchased, as the quantity required is not sufficient to justify the installation of an acid plant.   An acid plant will, however, be established in con-

nection with any large units that may be erected, the sulphur gases for the acid plant being obtained by the roasting of fine concentrates. The commercial success of the scheme in question, for the treatment of tailings or low-grade material of any kind, depends on being able to secure cheap acid. Where operations are conducted on a scale of several thousand tons per day, a large acid plant will be needed and the maximum economy in the manufacture of acid can be realized. Given roasting-furnace gases containing 6 per cent. or more of SO_2, the cost of making acid should not exceed $4 per ton of 60° Bé. acid.

The plan and elevation of the 80-ton leaching plant are shown in Figs. 1 and 2. The general arrangement does not differ materially from a cyanide plant for the treatment of gold ores. The plant will be used for the treatment of sand and slime tailings, which will be mixed in the proper proportions before entering the roasting furnace. Experiments have shown that a mixture of four parts sand tailings and one part slime tailings percolates without difficulty after roasting, the percolation rate after roasting being about three times as fast as before roasting, due to destruction of colloid in the furnace.

Fig. 3 shows a sectional elevation of the 20-ft. roasting furnace which is being constructed. The furnace is, in general, a six-hearth furnace of the MacDougall type. A seventh water-jacketed floor has been added for cooling the calcine preliminary to dropping it upon the belt conveyers leading to the leaching tanks.

There are two fire boxes, the flames from which pass over the third floor from the top. On the upper three floors the tailings are partly roasted and brought to a temperature of about 1,000° F., at which temperature they drop on to the fourth floor, where about 1 per cent. of salt is added. During its passage over the fourth, fifth, and sixth floors the remaining copper, as well as silver, is chloridized, the heat in the ore, which is kept from dissipating as much as possible, being sufficient for the reactions. A very small volume of air is drawn through the chloridizing hearths of the furnace by means of a fan, which exhausts into an absorption tower so as to catch any copper or silver which may have volatilized. The furnace is expected to have a capacity of at least 80 tons per day, on our material, and should produce somewhat better results, metallurgically, than the 16-ft. MacDougall in which the preliminary experiments were made.

Roasting Experiments in MacDougall No. 64.

This furnace was one of our regular MacDougall furnaces (No. 64), which we equipped with two fire boxes. These had grates 3 ft. by 2 ft. 6 in., and were arranged so that the flames could be made to
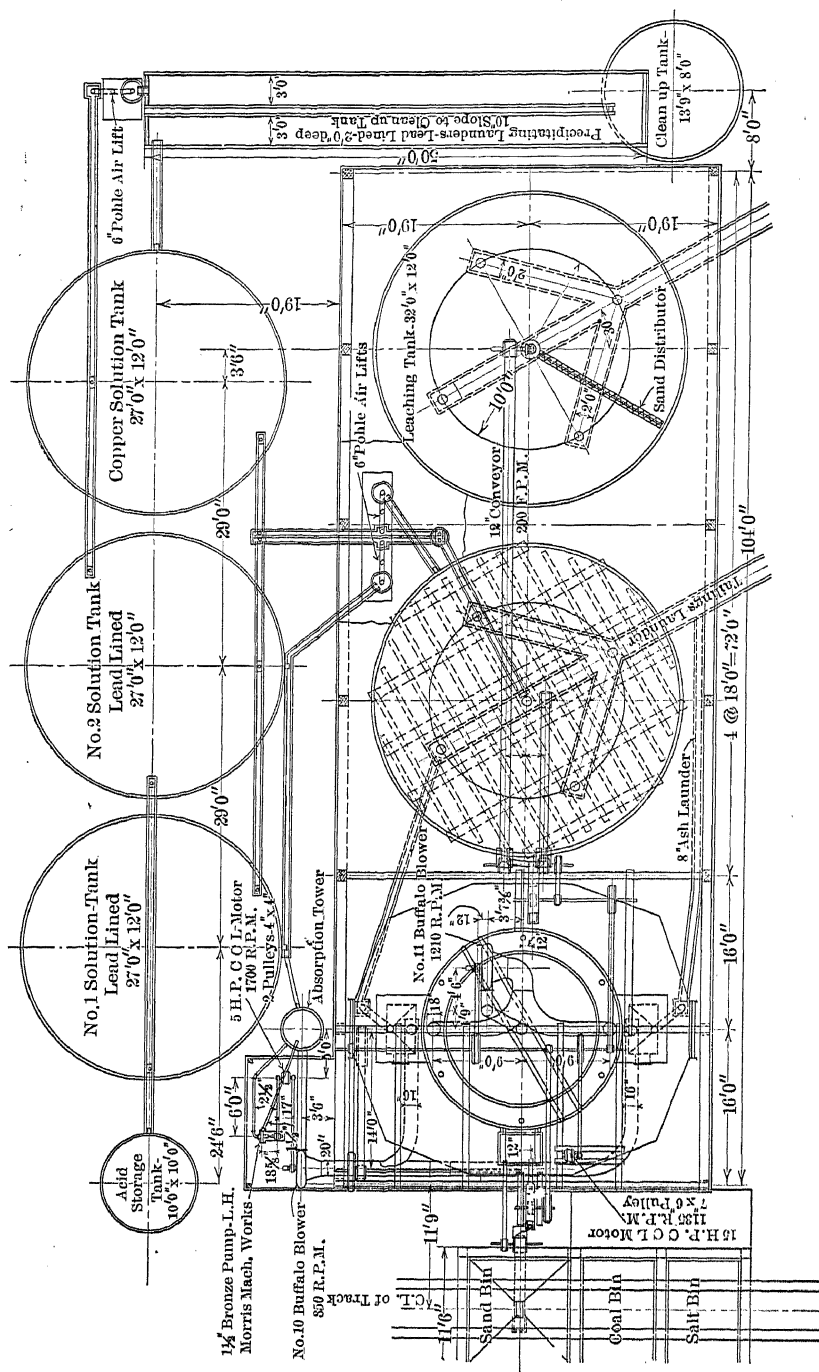
FIG. 1.—PLAN OF 80-TON LEACHING PLANT AT WASHOE REDUCTION WORKS, ANACONDA, MONT.
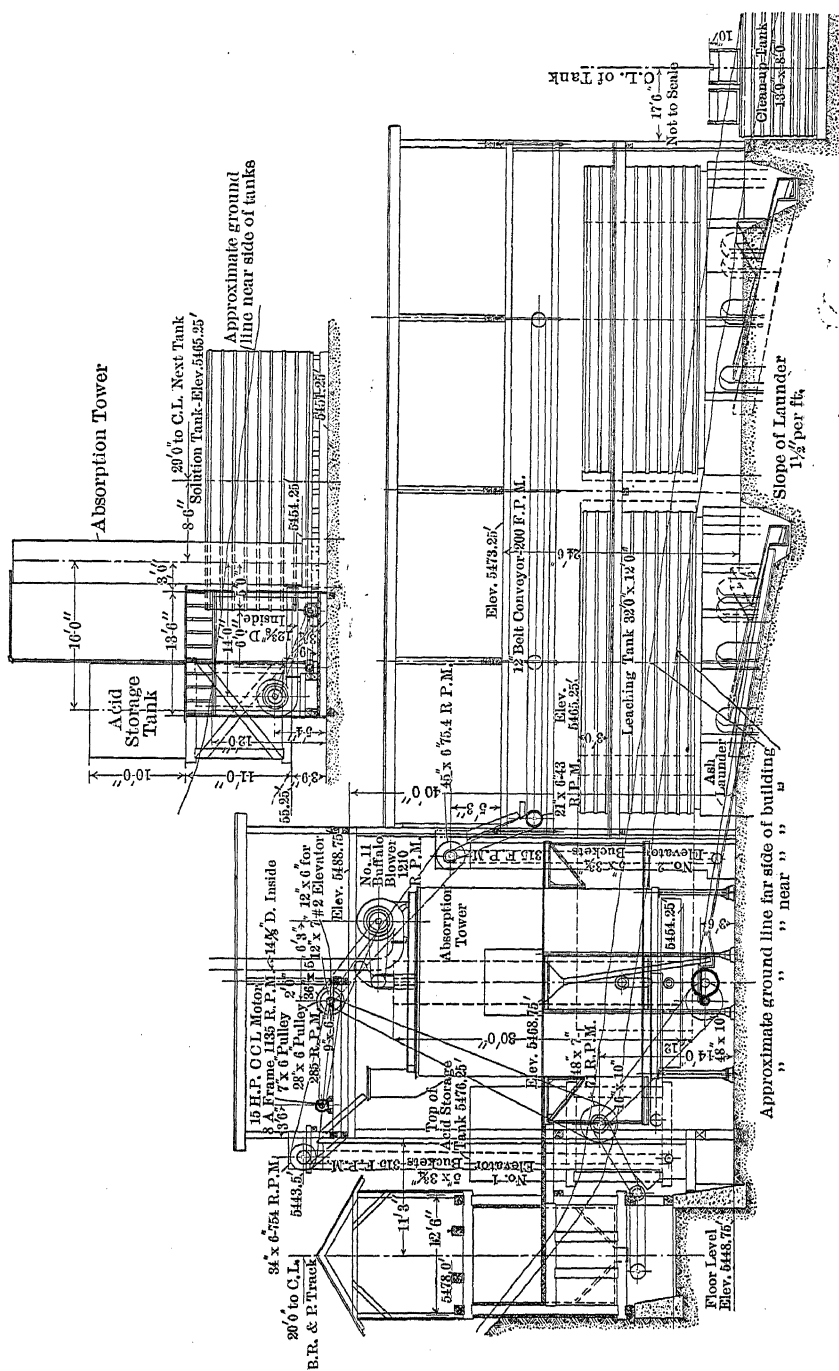
FIG. 2.—ELEVATION OF LEACHING PLANT.

enter on either the third or the fourth floor from the top.   Numerous observations were made to ascertain which was the better arrangement and it was finally decided that the third floor was better.   When the furnace was put on oxy-chloride roasting this was the only floor
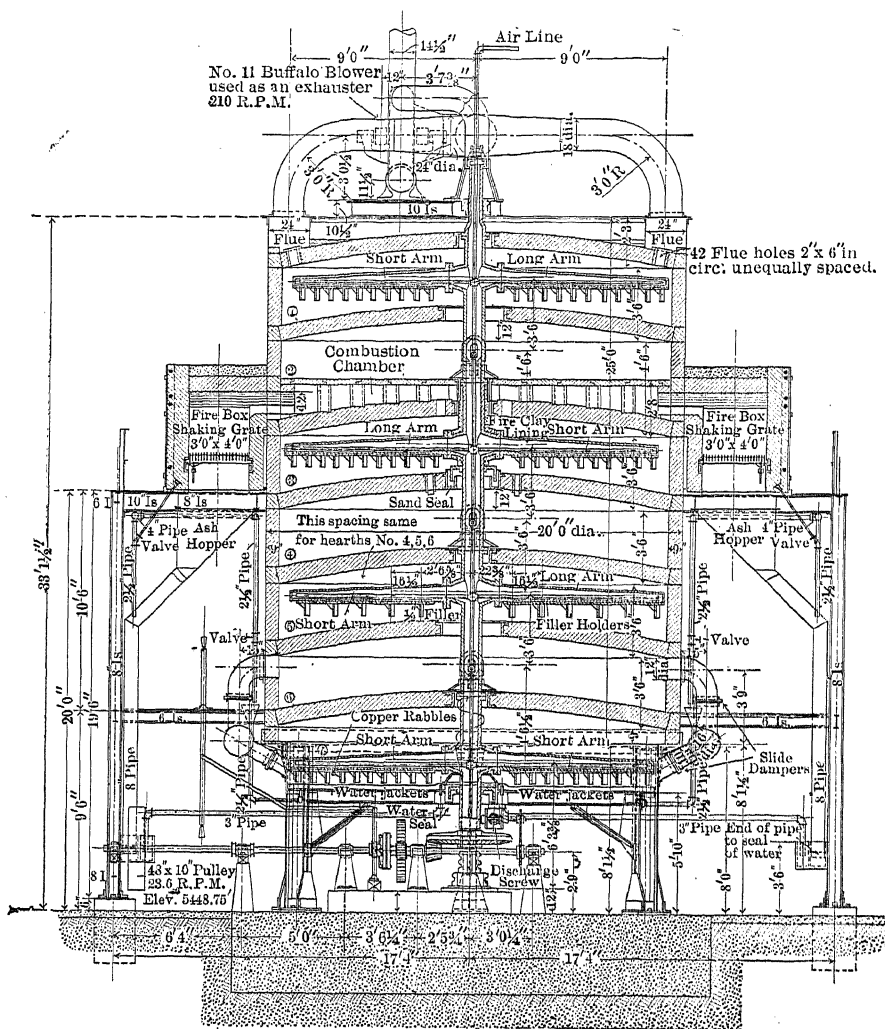
FIG. 3.—SECTIONAL ELEVATION OF 20-FT. ROASTING FURNACE.

that could be considered.   Experiments were also made with different speeds for the rabble arms.   The highest speed tried was one revolution in 23 sec.; the slowest, one revolution in 60 sec.   Most of the work was done with one revolution in 37 sec., but one revolution in

45 sec. gave just about the same results. Too high a speed pushes the material through the furnace too rapidly and does not give sufficient time for oxidation. Too slow a speed causes too thick a bed of calcine on the floors and does not permit the heat to penetrate readily.

When the furnace was used for oxy-chloride roasting, the rakes were removed from the short arm on each of the three lower floors. This caused a much thicker bed to form on these floors and gave only one-half as much stirring, all of which tended towards a better conservation of heat and better chloridizing.

The arms of the furnace and the shaft were air-cooled, except on the first and sixth floors, which were not cooled at all.

The furnace operated smoothly and very little labor was required on it, no barring off of accretions being necessary at any time. The firing was regulated by means of a pyrometer inserted over the fourth floor. It was found that the temperature could readily be kept within a few degrees of the desired point, which was 1,200° F. for oxidizing and 1,000° F. for oxy-chloride roasting.

Experiments were made with oxidizing, chloridizing, and oxy-chloride roasting. The last-named method gave the most satisfactory results, all things considered. The straight oxidizing was quite satisfactory, but only one-half of the silver was rendered soluble by it and the copper extraction was not so good as by the last method. The straight chloridizing was not satisfactory, owing to excessive loss of copper by volatilization and difficulty of catching same when mixed with fire-box gases. The salt consumption was, moreover, too high.

In carrying out the oxy-chloride roasting, the upper three hearths serve as oxidizing floors. Here the tailings are dried and heated to the proper temperature. All of the extra atom of sulphur in the FeS_2 is burned off and much of the FeS and Cu_2S are oxidized to oxides and sulphates.

When the ore drops from the third floor it is in excellent condition for chloridizing, which is done on the fourth, fifth, and sixth floors, by means of salt, which, to the extent of about 1 per cent. of the weight of calcine, is introduced on the fourth floor. Very little salt is required here, because most of the sulphur has been expelled, and no firing is needed on these floors, because they are well insulated and are kept hot by the heated ore constantly coming down. The volatilized copper is small in amount, because most of the Cu_2S has been converted into CuO before coming in contact with the salt and is, therefore, not chloridized, and whatever is volatilized can be easily caught by means of absorption towers, through which must pass only the small volume of gas from the lower floors.

The results obtained are given in Tables I., II., and III.

TABLE I.—*Test Period No. 1, Oxidizing Roast on Mill Tailings.*

Period of Test, June 4 to June 14, inclusive.

Date, 1912.	Feed. Wet Weight in Tons.	Feed. Dry Weight in Tons.	Calcine Dry Weight in Tons.	Coal Wet Weight in Tons.	Copper Remaining in Tailings After Leaching. As Cu_2S, Lb. Per Ton	As Ferrite. Lb. Per Ton.	As Total. Lb. Per Ton	Silver in Tailings After Leaching.	$Fe_2O_3 + Al_2O_3$ Dissolved. Per Cent.	Acid Consumption. Lb. H_2SO_4 Per Ton	Copper in Feed. Lb. Per Ton	Copper in Calcine. Lb. Per Ton.	Moisture in Feed. Per Cent.
June. 4	74.3	69.0	67.6	2 40	1 50	0.44	1 94	0 26	0.77	41.3	12.4	13.0	7.14
5	63 8	60 0	69.3	2 56	2 56	0 24	2 80	0.17	1.00	45 0	12.6	12.8	6.00
6	72 3	67.6	57 3	2.32	1 56	0 22	1 78	0.21	0.70	41.9	12.6	13.2	6.50
7	70.9	65 9	57 6	2.34	1.00	1.00	2.00	0.23	0 83	40.2	12.4	12.8	7.06
8	53 9	50 6	56.1	2.26	1 20	0 80	2.00	0 29	0.55	30.0	12.8	12.8	6.11
9	78 3	68.3	66.3	2.37	1 78	0 92	2.70	0.22	0.72	35.0	12 6	13.2	6.79
10	68.4	63 6	66.0	2 72	2.44	0.30	2.74	0 16	0.73	36.0	11.8	12 0	6.95
11	73.1	68.4	66 9	2 58	1.60	0 62	2.22	0.26	0.58	35 0	12.2	12 6	6.50
12	65.6	61.4	58 9	2.32	1 20	0 38	1.58	0.28	0.75	40.8	12.0	12.6	6.50
13	53.0	49.5	57.8	2.26	0.84	0 58	1.42	0.24	0.70	39.0	11.8	11.8	6 50
14	73.7	69.0	66 1	2 22	1.12	0.50	1.62	0.26	0.53	36.0	11.2	11.4	6.50
Total	742 3	683.3	689.9	26.35									
Averages	67 5	62.1	62 7	2.41	1.53	0.55	2.07	0.23	0.71	38 2	12.2	12.5	6.60

SUMMARY.

Pounds copper contained in feed   7,600
Pounds copper contained in calcine   7,850
Overage   250
Percentage of coal used to wet weight of feed 3.57
Percentage of coal used to dry weight of feed 3.88
Percentage of copper lost in tailings ...., ...   16.50
Percentage of copper recoverable 83.50
Percentage of silver lost in tailings 49.10
Percentage of silver recoverable 50.90
Cost of coal per ton of tailings (dry weight). (Diamondville coal at $4.90 per ton) .. $0 190
Cost of acid per ton of tailings (60° Bé acid at $3 54 per ton = $4.48 per ton H_2SO_4).... $0.082
Value of silver recoverable per ton of tailings $0 163

During the months of July and August, during which two of the test runs and practically all of the calcine for the leaching plant were made, our concentrator tailings assayed lower in copper than normally by about 0.05 per cent., or 1 lb. per ton.

There is no trouble whatever in maintaining a perfectly uniform calcine provided a uniform feed is maintained. Occasional "bad days" are generally due to the feeder failing to act properly and allowing the tailings to "run."

It will be noted that the flue-dust production was remarkably small, evidently having been less than 2 per cent. Even when a mixture of 5 parts tailings and 1 part slime was roasted, the weights of feed and calcine checked each other almost exactly after allowances were made for loss of sulphur, weight of salt, etc., showing that the flue-dust production must have been very small.

TABLE II.—*Test Period No. 2, Oxidizing Roast on Mill Tailings.*

Period of Test, July 8 to July 31, inclusive.

| Date, 1912. | Feed. Wet Weight in Tons. | Feed. Dry Weight in Tons | Calcine Dry Weight in Tons | Coal. Wet Weight in Tons. | Copper Remaining in Tailings After Leaching | | | Oz. Per Ton Silver in Tailings After Leaching | $Fe_2O_3 + Al_2O_3$ Dissolved. Per Cent. | Acid Consumption. Lb. H_2SO_4 Per Ton. | Copper in Feed. Lb. Per Ton. | Copper in Calcine. Lb. Per Ton. | Moisture in Feed. Per Cent. |
					As Cu_2S. Lb. Per Ton.	As Ferrite. Lb. Per Ton.	As Total. Lb. Per Ton.						
July													
8	62.7	58.8	50.6	2.54	1.43	0 12	1.55	0.22	0.75	43.3	11.2	10 4	6.23
9	62.7	58.5	70.3	2.33	1.10	0.76	1.86	0.22	0.61	42.6	10.8	11.6	6.70
10	82 6	76.8	60.3	2.39	1.00	0 68	1 68	0.21	0.70	43.0	10.8	11.2	6.97
11	63.2	59.0	61.5	2.39	1.44	0.96	2.40	0.24	0.75	38.2	12.2	11.8	6.62
12	75.5	70.5	64.0	2 37	1 56	1.30	2.86	0.18	0 75	38.0	11.8	13.0	6.62
13	75.0	70.5	65 7	2.39	2.20	0.74	2.94	0.14	0.83	36 5	11.8	12 8	6.00
14	64.7	61 5	60.4	2.31	1 60	0 76	2.36	0.15	0.95	39.5	10.8	11 6	5.00
15	75 8	72.3	67.2	2.52	2 46	0.72	3 18	0.21	0 95	39.0	11.0	11.6	4.59
19	65 0	59.7	49 4	2.39	1.86	0.74	2.60	0 34	0.79	44.0	11.6	12.4	8.20
23	78.6	75.1	70 4	2.80	2.00	0.94	2 94	0.22	0.76	39.0	10.8	11.6	4.39
24	75 2	71.9	59 8	2.40	1.80	0.98	2.78	0.26	0 82	39 0	11 8	12.8	4.37
25	66.1	63.3	70.2	2.29	1.50	1 02	2.52	0.24	0 75	37.0	12.6	14 0	4.30
26	51.2	48.7	39.4	2.16	1 44	0 90	2.34	0.28	0.76	42.3	12.2	14 0	4 90
27	43.4	41.3	48 2	2.14	1.30	0.88	2.18	0 27	0 90	45.5	11.4	12.2	4.88
28	55.2	52.0	39.3	2.22	1 20	0.82	2.02	0.23	0.78	45.0	10.4	11.8	5.71
29	65.2	62.2	70.2	2.25	1 74	0.52	2.26	0.23	0.79	38.0	11 2	11.6	4.59
30	61.2	58.7	48.4	2.11	1.20	0.80	2.00	0.25	0.58	40.8	10.6	12.2	3.99
31	64.6	60.4	62.8	2.07	1.76	0 62	2.42	0.24	0 80	36.7	12.0	12 2	6.51
Total...'...	1,187 9	1,121 2	1,058.1	42.07									
Averages.	65.9	62.3	58.8	2.34	1.59	0 79	2.88	0.23	0 78	40.4	11.4	12.2	5.58

REMARKS.—Furnace down from 16th to 17th, changing flame from 4th to 3d floor. Heating on 18th. Furnace down on 20th and 21st due to cave in flue.

SUMMARY.

Pounds copper contained in feed..	12,800
Pounds copper contained in calcine.........	12,900
Overage..................	100
Percentage of coal used to wet weight of feed.	3 55
Percentage of coal used to dry weight of feed.......	3.75
Percentage of copper lost in tailings	19.5
Percentage of silver lost in tailings	44.2
Percentage of copper recoverable..	80 5
Percentage of silver recoverable	55.8
Percentage of copper in calcine..................................	100.8
Cost of coal per ton of tailings (dry weight). (Diamondville coal at $4.90 per ton)......	$0.184
Cost of acid per ton of tailings (60° Bé. acid at $3.45 per ton = $4.48 per ton H_2SO_4)....	$0 086
Value of silver recoverable per ton of tailings...	$0.176

It is of the greatest importance in roasting for leaching to avoid overheating of the ore, since that would render insoluble a portion of the copper. According to Professor Hofman, this copper exists as ferrite. It is insoluble not only in the leaching solution, but also in concentrated nitric and hydrochloric acids, which makes its presence in the tailings very easy to overlook if care be not taken to effect the decomposition with hydrofluoric acid.

Following were the temperatures on the various floors of Furnace No. 64, and in flues and hopper:

TABLE III.—*Test Period No. 3, Oxide-Chloride Roast on Mill Tailings.*

Period of Test, Aug. 2 to Aug. 29, inclusive.

Date, 1912	Feed. Wet Weight in Tons.	Feed. Dry Weight in Tons.	Calcine Dry Weight in Tons.	Coal. Wet Weight in Tons.	Copper Remaining in Tailings after Leaching			Oz Silver Per Ton in Tailings after Leaching	Fe₂O₃+Al₂O₃ Dissolved Per Cent	Acid consumption, Lb H₂SO₄ Per Ton.	Copper in Feed. Per Ton. Lb.	Copper in Calcine Lb. Per Ton.	Moisture in Feed. Per Cent.
					As Cu₂S Lb Per Ton.	As Ferrite Lb Per Ton.	As Total. Lb Per Ton						
August.													
2	74.9	70.7	61.2	2.10	1.30	0.95	2.25	0.08	0.78	45.0	10.4	11.0	5.59
3	43.7	41.1	51.3	1.96	1.26	0.94	2.20	0.06	0.94	44.0	10.8	9.6	5.84
4	74.6	70.3	61.9	2.15	1.30	0.88	2.18	0.04	0.79	42.5	10.6	9.6	5.75
5	65.5	61.5	58.6	2.14	1.06	0.62	1.68	0.03	0.80	38.5	11.0	9.8	6.00
6	64.2	60.7	67.2	1.98	1.10	0.38	1.48	0.03	0.75	39.0	10.4	10.0	5.45
7	63.7	60.5	58.4	1.92	1.30	0.10	1.40	0.03	0.77	42.0	11.6	10.6	4.95
8	64.3	61.1	47.5	2.32	1.06	0.26	1.32	0.03	0.68	42.2	10.8	10.6	5.06
9	71.8	68.0	66.6	2.40	0.96	0.68	1.64	0.03	0.80	42.0	11.2	10.0	5.25
10	66.0	62.7	67.7	2.26	1.20	0.48	1.66	0.03	0.76	45.1	11.8	10.8	4.88
11	64.9	60.5	58.4	2.13	1.00	0.20	1.20	0.08	0.81	45.6	10.8	11.0	6.80
12	65.7	62.5	72.0	2.25	1.66	0.30	1.96	0.04	1.30	54.0	11.2	11.2	4.85
13	77.8	73.1	68.7	2.55	1.44	0.56	2.00	0.05	0.96	48.0	11.4	11.8	6.00
14	76.6	72.9	73.7	2.76	1.86	0.38	2.24	0.04	0.95	50.5	11.4	11.4	4.75
15	66.5	63.2	65.6	2.19	1.16	0.54	1.70	0.03	0.95	47.0	12.0	11.4	5.00
16	51.0	48.3	54.6	1.98	0.94	0.64	1.58	0.03	0.84	41.5	12.6	11.0	5.14
17	75.3	71.2	55.5	1.98	1.32	0.60	1.92	0.03	0.80	43.0	11.8	11.8	5.43
18	52.2	48.5	55.7	2.03	1.40	0.60	2.00	0.05	0.80	45.0	12.4	11.8	7.12
19	63.6	60.4	56.0	2.05	1.10	0.38	1.48	0.04	0.85	44.0	12.6	11.2	5.07
20	59.8	56.8	55.6	2.06	1.24	0.48	1.72	0.04	0.88	50.2 (salt'd)	11.6	11.2	5.00
21	53.6	50.9	55.3	2.12	1.14	0.54	1.68	0.08	0.79	45.0	11.6	11.0	5.00
24	75.3	71.6	64.8	2.27	1.26	0.38	1.64	0.08	0.80	47.0	11.4	11.4	4.88
25	74.0	70.9	64.7	2.56	1.54	0.54	1.98	0.08	0.85	46.5	11.4	11.4	4.23
26	73.4	70.4	75.8	2.46	1.70	0.18	1.88	0.03	0.90	45.0	10.6	11.0	4.08
27	64.2	61.1	57.9	2.10	1.10	0.50	1.60	0.02	0.80	47.0	10.0	10.0	4.80
28	42.5	40.0	38.0	1.99	0.84	0.62	1.46	0.02	0.81	47.0	11.2	10.0	5.97
29	54.1	50.5	48.6	2.06	0.84	0.48	1.32	0.02	0.80	43.5	11.0	10.0	6.57
Total..............	1,678.2	1,589.4	1,556.3	56.77									
Averages	64.7	61.3	59.9	2.18	1.23	0.51	1.74	0.035	0.84	45.0	11.3	10.8	5.29

REMARKS.—Furnace down from 22d to 23d, for replacing broken grate bars.

SUMMARY.

Pounds copper contained in feed... 17,950
Pounds copper contained in calcine......... 16,800
Pounds copper volatilized and in flue dust 1,150
Percentage of coal used to wet weight of feed......... .... 3.37
Percentage of coal used to dry weight of feed........ .... 3.56
Percentage of copper lost in tailings. 15 10
Percentage of silver lost in tailings....... .... 6.60
Percentage of copper recoverable...... . ... 84.90
Percentage of silver recoverable.................. .... 93.40
Percentage of copper volatilized and in flue dust... .... 6.41
Percentage of copper in calcine.... .. .. 93.59
Cost of coal per ton of tailings (dry weight). (Diamondville coal at $4.90 per ton)... $0.175
Cost of acid per ton of tailings (60° Bé. acid at $3.54 per ton = $4.48 per ton H₂SO₄).. $0.006
Cost of salt per ton of tailings (1 per cent. salt at $7 per ton) $0.070
Value of silver recoverable per ton of tailings.. .... $0 30

Temperatures.

Floor No.	Degrees Centigrade.	Degrees Fahrenheit.
1	71	160
2	282	540
3	693	1,280
4	571	1,060
5	450	840
6	326	620
Flue..	249	480
Calcine in hopper..............................	277	530

There were drawn through the furnace per 24 hr. a little more than 6,000,000 cu. ft. of air, as compared with 5,500,000 cu. ft. for our regular furnaces. Much of this came through the lower floors and tended to cool these and remove the chloridizing fumes, thus detracting from their efficiency and from the heat economy of the furnace. This will be remedied in the furnace now under construction.

Everything entering and leaving the furnace was carefully sampled and the amount of recoverable copper in the unground calcine was determined for each shift by treating a 200-g. sample with 150 cc. of a leaching solution, containing 8 per cent. of H_2SO_4 and 10 per cent. of NaCl, at a temperature of about 85° C. for 2.5 hr.

Leaching Experiments on Calcine from MacDougall No. 64.

A small leaching plant capable of treating about 20 tons at a charge was erected, for the purpose of ascertaining what results could be obtained on the roasted tailings on a working scale. The plant is shown in plan and photograph in Figs. 4 and 5. The copper was dissolved by percolation with solutions containing sulphuric acid and common salt, and precipitated by means of sulphuretted hydrogen made by the action of dilute sulphuric acid on an iron sulphide matte, the resulting precipitate being collected in a small filter press.

There were four solution tanks. The dimensions of the largest were 10 by 10 by 7 ft. and it contained the strong, or No. 2, solution, which carried about 6 per cent. of H_2SO_4 and 10 per cent. of NaCl. There were three smaller tanks, of dimensions of 7 by 7 by 7 ft. One of these held the weak, or No. 1, solution, which carried about 3.5 per cent. of H_2SO_4 and 10 per cent. of NaCl. The other two were tanks for wash water, to be used for washing the tailings after leaching.

There was another large tank, 10 by 10 by 7 ft., which was used for leaching purposes. This tank had a false bottom, which was covered with cocoa matting.

The precipitation tank was also 10 by 10 by 7 ft., was equipped with an agitator, and had a system of lead pipes in the bottom for carrying and distributing the H_2S gas through the solution.

A small tank for containing excess copper solution was built, but it was not found necessary to use it. In case, at any time, the precipitation of the copper sulphide was carried too far, and the solution absorbed excess H_2S, which made it unfit for leaching purposes, this copper solution was to be drawn into it, to use up this excess H_2S.

All circulation of solution was accomplished by means of an air lift, built of 4-in. lead pipe, drawing from a sump 4 by 4 by 4 ft.; and
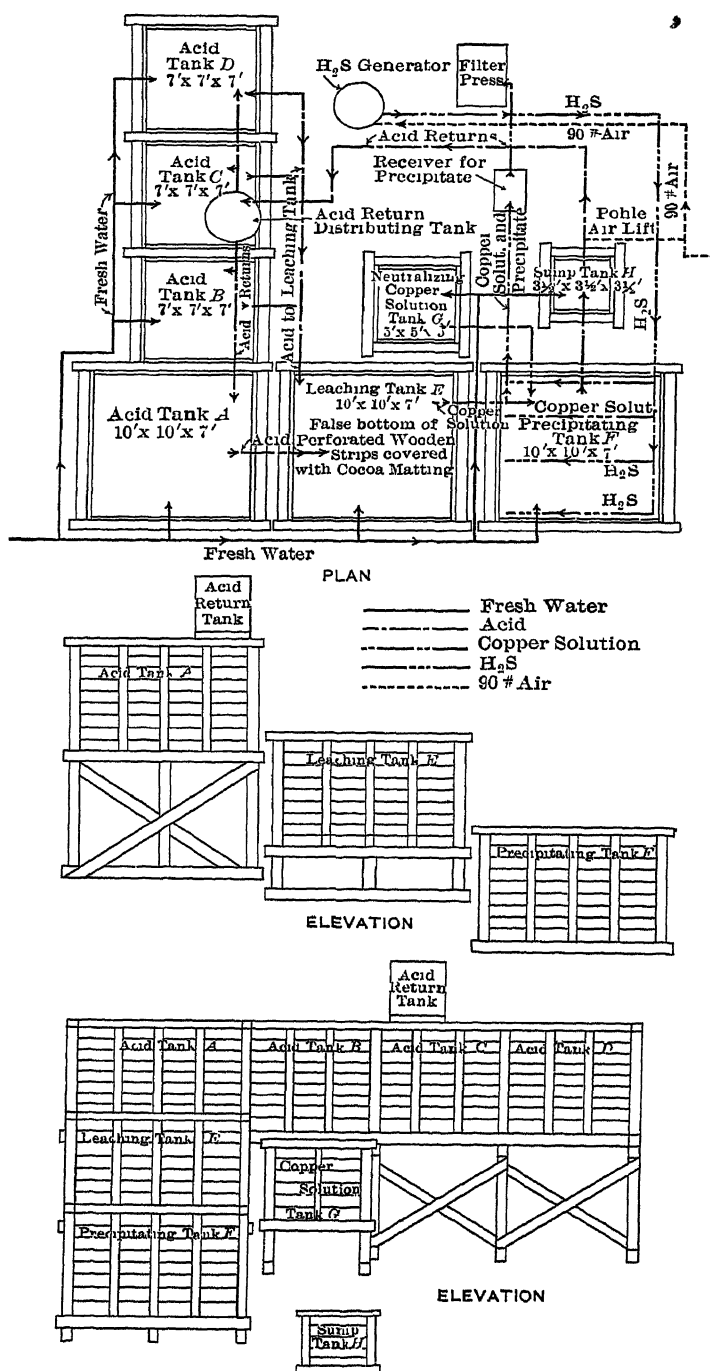
FIG. 4.—PLAN AND ELEVATIONS OF EXPERIMENTAL LEACHING PLANT.

operated by air at 15 lb. pressure per square inch.   This air lift was very satisfactory and never gave any trouble.   It discharged into a distributing box above the solution tanks, and, by means of valve manipulation, could be made to lift solution from any tank to any other tank in the plant.   All the pipes which carried acid or copper solutions were of lead.   During the time of operation, about three months, these lead pipes and fittings stood up very well and showed very little corrosion, if any, while any copper or brass fittings used had a very short life.   Large pinch cocks on short pieces of rubber hose were used as valves.   They were quite satisfactory.

The H_2S generator was an iron drum built to stand 90 lb. pressure per square inch, lead lined, and of an inside diameter of 2 ft., and an inside height of 4 ft.   The iron sulphide was introduced through a hand hole in the top.   There were both steam and air connections on the drum.
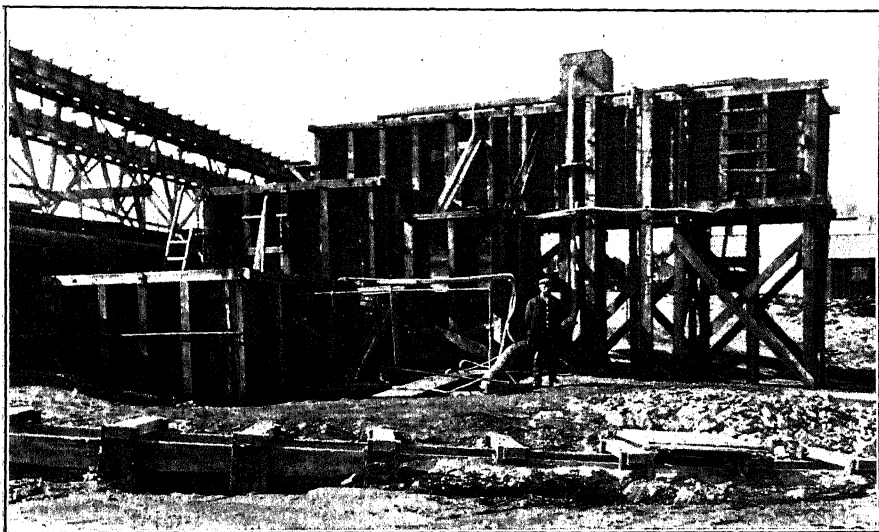
FIG. 5.—VIEW OF EXPERIMENTAL LEACHING PLANT.

The sulphuric acid was introduced in a concentrated condition, through an iron pipe, the container being an iron barrel set about 20 ft. above the generator.   The gas was carried through a lead pipe to the pipes mentioned above, which lay in the bottom of the precipitating tank.   These pipes had small holes in them, spaced at regular intervals, for the escape of the gas.

In the bottom of the generator was a hole closed by a bronze plug. When the action ceased, this plug was removed, and the 90 lb. air pressure applied.   This blew out all of the water and ferrous sulphate, and the generator was ready for charging again.

The filtration of the copper sulphide was done in a small wooden filter press. The inside dimensions of the frames were 12 by 12 by 1⅝ in. Ordinary filter paper was used as filtering medium with heavy canvas for a backing. The copper sulphide was allowed to settle over night and the clear solution then drawn off, so that the material for the filter press was quite dense. This thickened material was drawn into a lead-lined drum of the same dimensions as the generator, and forced through a 0.5 in. lead pipe to the filter press, with 90 lb. air pressure.

The scheme of operation is very simple, depending on the two chemical formulæ:

$$H_2SO_4 + CuO = CuSO_4 + H_2O.$$
$$CuSO_4 + H_2S = CuS + H_2SO_4.$$
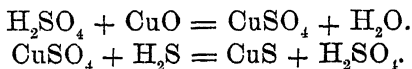
The weak solution, carrying about 3.5 per cent. of H_2SO_4 and 10 per cent. of NaCl, was run on the tailings first, to get out the bulk of the copper. This solution was the only one precipitated, and about 18,000 lb. of it were used to about 24 tons of calcined tailings. The same amount of strong solution, carrying 6 per cent. of H_2SO_4 and 10 per cent. of NaCl, was then used and returned, to be used as the No. 1, or weak, solution in the next leach.

The No. 1 solution remained in contact for 14 hr.; the No. 2 solution for 72 hr. The values were washed out with a weak solution and then with fresh water.

The final tailings were sluiced out of the tank with a 2-in. hose, and carried to waste by means of a launder carrying a 3-in. stream of water.

The solution to be precipitated was heated to 45° C. and the H_2S gas turned in. It is very necessary that there be a good agitation during precipitation. After the amount of copper in solution was lowered to about 0.06 per cent., the precipitate was allowed to settle over night and the clear solution decanted and lifted to the strong solution tank, to be used for the next leach. The solution was brought up to strength in both acid and salt during precipitation.

It was found that no salt was required to be added to the solutions, as they picked up salt from the tailings. When the clear solution was decanted as close to the precipitate as possible, the remaining pulp was ready for filtering. It was heated and mixed, and drawn into a lead-lined drum, and forced through the filter press under 90 lb. air pressure, giving the final product—a black sulphide of copper, containing some impurities.

The little plant proved that fully as good results could be obtained by leaching on a moderately large scale as by the method used on the daily samples in the laboratory, previously described.

The summarized results of two months' operations are as follows:

TABLE IV.—*Copper and Silver Recoveries.*

	Copper. Lb per Ton.	Silver. Oz. per Ton.
In feed to roasting furnace,	11.30	0.53
In calcine to leaching plant,	10.40	0.46
In volatilized but recoverable,	0.90	0.07
In tailings from leaching plant,	1.70	0.05
Percentage recoverable copper,	85.4	
Percentage recoverable silver,		91.1

Miscellaneous Data.

Percentage of dissolved copper in No. 1 solution, ... 91.1
(Treatment time, 14 hr.   Acid consumed per ton calcine, 13.6 lb.
H_2SO_4.)

Percentage of dissolved copper in No. 2 solution, ... 9.0
(Treatment time, 72 hr.   Acid consumed per ton calcine, 19.1 lb.
H_2SO_4.)

Pounds $Fe_2O_3 + Al_2O_3$ dissolved per ton of calcine 7.7

The analysis of the tailings treated was as follows:

Cu. Per Cent.	Ag. Oz.	Au. Oz.	SiO_2. Per Cent.	Fe. Per Cent.	S. Per Cent.	Al_2O_3 Per Cent.	CaO. Per Cent.
0.60	0.55	0.002	82.2	1.9	2.2	9.4	0.6

Tables V. and VI. give the total sulphur in feed and calcine and the sulphate sulphur in the calcine from No. 64 MacDougall during Test Periods 2 and 3, respectively.

TABLE V.—*Total Sulphur in Feed and Calcine, No. 64 MacDougall, and Sulphate Sulphur in Calcine.*

Date. July.	Feed. Total Sulphur. Per Cent.	Calcine. Total Sulphur. Per Cent.	Calcine. Sulphate Sulphur. Per Cent.
8	2.30	0.52	0.26
9	2.21	0.33	0.20
10	2.32	0.42	0.22
11	2.67	0.48	0.25
12	2.36	0.42	0.22
13	2.42	0.49	0.40
14	2.50	0.55	0.30
15	2.60	0.59	0.21
16	2.50	0.55	0.33
17	2.50	1.21	0.18
18	2.70	0.59	0.30
19	2.85	0.74	0.38
20	2.50	0.62	0.42
21	2.52	0.59	0.22
22	2.65	0.59	0.30
23	2.60	0.56	0.32
24	2.50	0.56	0.36
25	2.52	0.53	0.27
26	2.42	0.45	0.23
27	2.35	0.30	0.28
28	2.20	0.32	0.26
29	2.30	0.45	0.25
30	2.28	0.59	0.26
31	2.50	0.51	0.29
Average,	2.43	0.54	0.28

TABLE VI.—*Total Sulphur in Feed and Calcine, No. 64 MacDougall, and Sulphate Sulphur in Calcine.*

Date. August	Feed. Total Sulphur. Per Cent.	Calcine. Total Sulphur. Per Cent.	Calcine Sulphate Sulphur. Per Cent
2	2.47	0.50	0.34
3	2.30	0.48	0.37
4	2.20	0.47	0.34
5	2.37	0.46	0.34
6	2.42	0.50	0.40
7	2.47	0.51	0.40
8	2.47	0.53	0.45
9	2.54	0.51	0.37
10	2.40	0.48	0.40
11	2.54	0.50	0.40
12	2.61	0.59	0.32
13	2.70	0.51	0.34
14	2.70	0.52	0.40
15	2.60	0.52	0.38
16	2.50	0.51	0.41
17	2.60	0.52	0.46
18	2.50	0.55	0.49
19	2.55	0.52	0.44
20	2.70	0.55	0.49
21	2.60	0.56	0.48
22	2.40	0.56	0.46
23	2.60	0.56	0.34
24	2.60	0.55	0.44
25	2.50	0.51	0.40
26	2.50	0.53	0.41
27	2.40	0.53	0.48
28	2.10	0.48	0.42
29	2.20	0.44	0.37
Average,	2.48	0.52	0.44

Iron matte for the generation of H_2S was made by fusing with limestone and oxidized iron ore a gold-bearing pyrite ore of the following analysis:

Cu. Per Cent.	SiO_2. Per Cent.	Fe. Per Cent.	CaO. Per Cent.	S. Per Cent.
1.2	23.0	31.6	1.4	35.9

The resulting matte analyzed as follows:

Cu. Per Cent.	SiO_2. Per Cent.	Fe. Per Cent.	S. Per Cent.
2.23	0.8	61.3	31.1

It decomposed readily with dilute acid and yielded practically the theoretical quantity of H_2S. About 20 per cent. excess of H_2S was required in precipitating (under conditions where all of the gas entering the solution was absorbed) in order to make up losses due to ferric salts in the copper solution. The acid for generating H_2S was,

of course, regenerated in the leaching solution.   The precipitate of copper sulphide analyzed as follows:

Cu Per Cent.	Ag. Oz.	Au. Oz.	S. Per Cent.
58	69.5	0.04	29.0

The press cakes were firm and easily handled, but contained, nevertheless, 48 per cent. of moisture.

Conclusion.

There is a good deal to be said in favor of a leaching process as a finishing process following water concentration, since this method is unquestionably capable of giving a greater recovery of copper than any other system in general use at the present time.

The treatment of high-grade (say 3 per cent.) easily concentrated material directly in a leaching plant is, in my opinion, not advisable, owing to greater consumption of acid and chemicals, smaller furnace capacity, and greater tailings loss.   In order to prepare material for treatment by roasting and leaching, it is necessary to crush it to at least 15 mesh.   Whatever copper is set free down to this point should be taken out as high-grade concentrates and reduced by smelting.   It is when we come to the fine grinding and concentrating part of the mill that the trouble commences.   Here we are between the devil and the deep sea.   We must grind fine in order to free the mineral, but in doing so we cannot avoid the production of a large percentage of slimes on which the concentrating machines now available can make but a very unsatisfactory recovery.

The manufacture of sulphuric acid at a smelting plant can be cheaply carried on and offers no special difficulties.   From the "smoke nuisance" standpoint, it is an advantage to the smelter to be engaged in this industry.   Roasting and leaching, under conditions as they exist at Anaconda, can be readily done on any scale.   In short, the entire matter resolves itself into a question of costs.

We have reason to believe that tailings material at Anaconda can be roasted and leached for not more than 70 cents per ton, which means that copper can be made from tailings yielding only 10 lb. per ton for 7 cents per pound.   The recovered silver will pay for shipping, marketing, and refining plus about 0.5 cent, making the net cost of the copper about 6.5 cents per pound.   These costs are made possible by the fact that the material in question is already mined, crushed, and sized, and can, of course, be obtained only by making cheap acid from roaster gases and operating on a very large scale, say not less than 3,000 tons per day.

DISCUSSION.

J. C. DICK, Salt Lake City, Utah :—I would like to ask Mr. Laist why a 10 per cent. salt solution was used.

MR. LAIST :—The reason for using salt in the solution is to get the silver. The solution of dilute sulphuric acid has no solvent action on the silver even after roasting, and the addition of the salt is made primarily to dissolve the silver chlorides.

MR. DICK :—I would like to ask also if you made any experiments as to grinding fine your calcines, rather than leaving the coarse material.

MR. LAIST :—We have not made any experiments along those lines except in the laboratory, but the objections which we have to grinding are that the difficulties of roasting are very much greater when roasting finely ground material than when roasting coarse material, and the difficulties of leaching are somewhat increased; the percolating system is simpler and easier to manipulate, particularly when acid solutions are used, so that we prefer from our experience not to grind.

STUART CROASDALE,[*] Denver, Colo. (communication to the Secretary [†]):—In order to meet the conditions involved in his ore-treatment problem, Mr. Laist has developed a very interesting and apparently successful metallurgical hybrid which he calls the oxychloridizing roast. The lower floors of the MacDougall furnaces are made to accomplish what used to be done on the "cooling floor" in the old days of salt roasting and "hypo" or brine leaching of silver ores.

His experience with the straight chloridizing roast recalls my own experience, of 10 to 15 years ago, while experimenting on the volatilization of metals as chlorides. I found that the highest loss by volatilization occurred when the salt and sulphur were present in quantities not only to combine theoretically with each other but also with the volatile base metal, like copper or lead, to form the normal sulphide and chloride of that metal. For example, an ore containing 2 per cent. of copper, if finely ground and roasted with 1 per cent. of sulphur and 4 per cent. of salt, would yield almost complete vola-

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tilization of the copper, while if roasted with the same amount of sulphur and 2 per cent. or 8 or 10 per cent. of salt the loss by volatilization would not be nearly as high. The same results would occur if the theoretical amount of salt was maintained and the quantity of sulphur was thrown out of proportion.

Carbonates, oxides, and silicates of copper could be readily chloridized and volatilized by the addition of sulphur as iron pyrite, and roasting with salt. Sulphide ores could be roasted " dead " and the copper could be volatilized by adding the theoretical quantity of salt and sulphur and again roasting at the proper temperature. Chalcocite and antimonial ores gave less loss by volatilization than chalcopyrite and the higher sulphides of copper.

Silver losses by volatilization are high if the volatile chlorides of the base metals are present, such as copper and lead. They are also much higher, at a given temperature, if an excess of air is admitted to the furnace and the ore is constantly rabbled, than they are when a minimum amount of air is admitted, or the furnace is closed and little or no rabbling is done.

The volatilization of copper begins at a dull red heat, or about 1,200° F.—perhaps below this point.

In a chloridizing roast, however, the dust losses cease as soon as chemical action begins, which is in the neighborhood of 1,000° F.

I experienced the same difficulty that Mr. Laist mentioned in trying to condense or collect copper chloride fumes when mixed with fire-box gases. These fumes seem to act much the same as zinc vapor under similar conditions. I drove the mixed gases into a condensing tower filled with fine gravel to a height of 8 ft. and covered with 3 in. of sand. This was sprayed with an excess of water. The copper chloride fumes came through it all, perfectly cold, but apparently untouched by the water. I have often thought that Dr. Cottrell might be able to collect them electrostatically when mixed with the products of combustion, but I have never tried it.

On the other hand, if these fumes are kept free from the products of combustion they are readily condensed by ordinary methods.

Mr. Laist brings out another point which I think is very important where roasting precedes the leaching of ore or tailings, and that is the formation of ferrites. This is particularly noticeable in roasting zinc ores.

I am somewhat surprised at the comparatively high acid consumption on his roasted material. I presume this is due to the ferrous and basic salts of iron formed by roasting at this temperature.

Carbonate ores with a granitic or porphyry gangue, that have been oxidized in place and in which no leaching has occurred, will contain considerable soluble iron and alumina. The iron will usually pass into solution as readily as the copper and the alumina soon follows, even in weak acid solutions. Since Mr. Laist employs an oxidizing roast as a preliminary operation, I believe he would find it advantageous to carry it to a higher stage of oxidation and render the iron and alumina more insoluble. This should not affect the chloridization of silver in any way.

The precipitation with hydrogen sulphide is an interesting development for the treatment of Anaconda tailings and we shall be glad to watch its continued use on a larger scale. At the present time precipitation is the most difficult part of the leaching problem to solve economically, especially on an Arizona desert, where the hydrometallurgy of copper is likely to reach its greatest development in this country.

W. McA. JOHNSON, Hartford, Conn. (communication to the Secretary*):—I have read and re-read with pleasure Mr. Laist's paper on Roasting and Leaching Tailings at Anaconda, Mont. Mr. Laist's work has to my mind a distinct practical and commercial bearing on metallurgy and on the metal business, and so far as my powers allow me to I wish to consider briefly this phase of the subject.

Having been engaged for a considerable period in investigating new processes for separating copper-nickel matte for the old Orford Copper Co., and later having studied many processes for the treatment of zinc-lead ores, I have had, I believe, a certain sort of special experience in the development of new processes.

The valuable part of the knowledge derived from this experience can be summed up best as follows:

The ratio of the experimentally expended dollar to sum of the profits made from success, should it come, must be a maximum. In most cases, of course, this efficiency factor of the development of a new process is zero, for most attempts fail and in many successful cases the amount of money wasted is necessarily large. For instance, even Sir Henry Bessemer had to build 17 converters before his process and apparatus were practicalized. Now in the last analysis all this money apparently wasted in experimenting is a charge against the cost of making metal just as the money lost in prospecting and wasted in wild-cat mining ventures is a charge against the cost of making metal. Parenthetically let me state that in view of the industrial

absolutism of metal through modern civilization the price of metals does not average high enough when we consider the great amount of honest effort expended in finding mines, and new processes to make refractory ores available, and in the actual mining of the ore, smelting, and refining the metallic products thereof.

But in Mr. Laist's experimental work, as far as I can judge, a great deal of progress has been made simply as a result of frank discussion and of good counsel, and by the exercise of care and foresight; in short, a great deal of experimenting has been done by talking and on paper.   For this reason I think that Mr. Laist's work is greatly to be commended.

Let us now consider his conclusion that the treatment of 3 per cent. concentrates is not advisable, but that the process is profitable and commercial on treating concentrates containing 0.5 per cent. of copper.   This is, to my mind, in exact accord with a general broad principle of metallurgy that a series of processes or sub-processes, each performing its function at a maximum efficiency, makes the cheapest final operation and, of course, the cheapest metal.   For instance, in roasting copper-nickel matte down from 22 per cent. of sulphur to 0.02 per cent. of sulphur from three to five operations are necessary, each one taking out the part of the sulphur for which the special operation is best fitted.   A simpler example of this is the crushing of ores in several stages by crushers, rolls, and fine-grinding mills.   If, therefore, we divest the question of its non-essentials and regard only the essentials we can see that Mr. Laist's process fits in the general metallurgy of Anaconda in a manner similar to the way that the cyanide process fits into the metallurgy of gold, where part of the gold is collected by mercury in the old-fashioned stamping process and where the product from the stamp mills is concentrated and the concentrates sold, and finally where the tailings are cyanided.

If Mr. Laist's proposal can be criticized at all it is that the capital cost of the demonstrating plant at Anaconda seems much higher than is really necessary and that the demonstrating plant is "over-designed."   These are good faults in new work and such criticism can well be regarded as captious.   If all the costs are figured on the same basis as $4 for 60° B. sulphuric acid, out of 6 per cent. dust-free SO_2 gas, the financial returns will be gratifying, through chamber acid would be cheaper and as good.   It also seems apparent that some continuous process for the leaching of the chloridized product must be finally used when the enormous tonnages that are rolling in at the Anaconda plant are treated.   The means for precipitating the copper is, of course, susceptible of great improvement.

My regard for the work is high and my admiration genuine. Nor will any one adduce from this discussion of mine the argument that because the work has been done cheaply and efficiently copper should sell for less than 16 c. per pound.

R. C. CANBY, Miami, Ariz. (communication to the Secretary *):— Mr. Laist has shown the economic possibility of roasting and leaching at a figure which heretofore would have been considered an impossibility. Since having seen the conditions under which he roasted the 5,000 tons of tailings and having had the benefit of his personal description of his work, I have felt convinced that Mr. Laist and the Anaconda Company have inaugurated a new era in this branch of work.

A comparison of the temperatures of the different hearths of his No. 64 MacDougall roaster impressed me with the possibility of attaining even greater fuel economy by adopting the practice of Messrs. Sörenson and Barker[1], and I have had in mind trying a furnace with the cast-iron hearths above the fired hearth, so as to gain the advantage of the convection of the heat, so that instead of a difference of 740° F. between the fired hearth and the one next above it, they should be more nearly equal. The manner of construction, with reinforced concrete for the fired hearths and hearths below, with less radiation from the walls than from the standard MacDougall furnace, would seem ideal.

There is one point which I hope will appear more fully in the discussion of Mr. Laist's paper, and of Mr. Wedge's paper read before the Boston Section, and that is as to how great is the advantage of the muffle type of furnace, which has seemed to me prohibitive in its fuel requirements. If the heat upon the approaching hearths were brought more nearly to 1,000° F., the fired hearth would undoubtedly require a less intense flame and there would be less danger of forming copper ferrite.

For using oil fuel upon such low-temperature roasts I propose a special burner for maintaining the ignition and economy of combustion, since in the Southwest oil is the more economical fuel.

Cast-iron flukes were used in cylindrical roasters at the Argentine plant, roasting 42 per cent. sulphur Congress concentrates for producing SO_2 gas for the Hunt and Douglas process, and I found cast iron gave better satisfaction than any other mode of construction.

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[1] *Engineering and Mining Journal*, vol. xcv., No. 26, p. 1273 (June 28, 1913).

Increasing the Efficiency of MacDougall Roasters at the Great
Falls Smelter of the Anaconda Copper Mining Co.

BY FRANK R. CORWIN AND SELDEN S. RODGERS, GREAT FALLS, MONT.

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I. INTRODUCTION.

SINCE the first installation of MacDougall roasters at the Great Falls
smelter of the Anaconda Copper Mining Co., the capacity of the fur-
naces has been more than doubled. During the first nine or ten years
of the operation of the MacDougall department the tonnage treated by
the furnaces remained practically unchanged. Experimental work
carried out in 1906, however, showed that increasing the draft of the
furnaces would bring about an increase in capacity. In 1909 the
MacDougall department was connected to the new smelter stack and

flue system, and the stronger draft caused by this change increased the capacity of the furnaces. The change in draft conditions made it necessary to operate differently, and systematic experimental work was started to adapt the furnaces to the new conditions. This experimental work extended over a period of several years, and resulted in very largely increasing the capacity of the MacDougall furnaces, decreasing the percentage of flue dust, and also brought about other changes and improvements tending to raise the efficiency of the MacDougall department. These changes, of course, resulted in reducing operating expenses. The improvement was brought about without building any new MacDougalls or enlarging the old ones, and was the result of the thorough study of the MacDougalls and numerous tests.

It is the purpose of this article to give a brief account of the experimental work and to show how the increase in efficiency was brought about.

Part of the experimental work described in this paper was carried out by the writers, while in the employ of the Anaconda Copper Mining Co., under the direction of M. W. Krejci, Metallurgist, and Arthur Crowfoot, Chief Sampler. Complete records of all of the MacDougall testing work have been placed at our disposal through the courtesy of A. E. Wheeler, Superintendent.

II. Brief History of Present MacDougall Plant.

The first change from the Brueckner cylinders used for roasting concentrates at Great Falls was the construction of a MacDougall roaster of the Herreshoff type. This furnace was first started on June 15, 1898. It had eight hearths, was lined with fire brick, and had an air-cooled central shaft, but no provision was made for cooling the rabble arms. It was operated for a while, and accomplished the work of roasting, but ran so hot that the rabble arms became twisted and distorted out of shape. To prevent the furnace from becoming too hot a device was then installed to draw the heated gases away from the hearths. This consisted of a pipe connected to alternate hearths of the MacDougall furnace and also connected to the flue. It was equipped with dampers, so that the hot gases could be drawn away from any one of the alternate hearths directly into the flue.

This first furnace not proving altogether a success, the cooling device was eliminated and the furnace shut down to be reconstructed. When rebuilt it was what is known as the Evans-Klepetko type of MacDougall furnace, having six hearths and water-cooled central shaft and rabble arms. Another MacDougall furnace was built at the same

time. These two furnaces are still in use, and are Nos. 8 and 9, respectively, of the present 22 MacDougall furnaces. Except for a few minor changes, such as substituting solid arms for the water-cooled arms on the first hearth, the removal of the branch pipes at first used inside of the rabble arms to distribute the cooling water, and changes in design of the rakes, rake frames and rabble arms, they still remain essentially the same as when built in 1899. Six more furnaces, now known as Battery III., were built and started in November, 1899. In the spring of 1900 four more MacDougall furnaces were constructed, making with Nos. 8 and 9 the present Battery II. of six furnaces. In 1901, Battery IV., consisting of six furnaces, was built, and in February, 1903, the four furnaces comprising Battery I., these last four furnaces completing the equipment of 22 MacDougall furnaces.

III. First Experimental Work on Increasing Capacity of Furnaces.

The composition of the concentrate treated when the MacDougall plant was first started was approximately as follows:

Cu. Per Cent.	SiO₂ Per Cent.	Fe. Per Cent	Al₂O₄ Per Cent.	Zn. Per Cent.	S. Per Cent.
13.3	24.2	25.6	4.6	0.9	30.0

When first built, the MacDougall furnaces had a rated capacity of 40 tons of raw wet concentrate containing about 35 per cent. of sulphur, roasted down to 7 per cent. of sulphur, per furnace per day. In actual practice, however, the capacity of the furnaces fell somewhat below this, averaging roughly around 35 tons of total cupreous material per furnace per day. The first experimental work looking towards an increase in the capacity of the furnaces was carried out by J. H. Klepinger, in June and July, 1906, the primary object of the tests made by Mr. Klepinger being to determine whether an increase in the draft of the MacDougalls would increase their capacity. These tests were made previous to the building of the new smelter stack.

MacDougall furnace No. 2 was selected for the test, on account of its favorable location. To increase the draft on No. 2 furnace, a No. 8 Sturtevant exhauster was connected to the top of the furnace, as shown in the accompanying sketch from Mr. Klepinger's report, (Fig. 1), the gas being discharged into the main MacDougall cross flue. When the exhauster was in use the slides in the regular draft necks were closed. When for any reason the exhauster was stopped these slides were drawn out, allowing the furnace to operate under

natural draft. Several tests were made on No. 2 furnace, first work-
ing under natural draft, and later with the exhauster in operation.
During the tests a careful record was kept of the weights of material
charged to the furnace, samples were taken of the concentrate charged
and calcine produced, and observations taken to determine the draft,
and the temperature and volume of the discharged gases. A brief
summary of part of the principal data obtained, taken from Mr. Klep-
inger's report, is tabulated herewith.
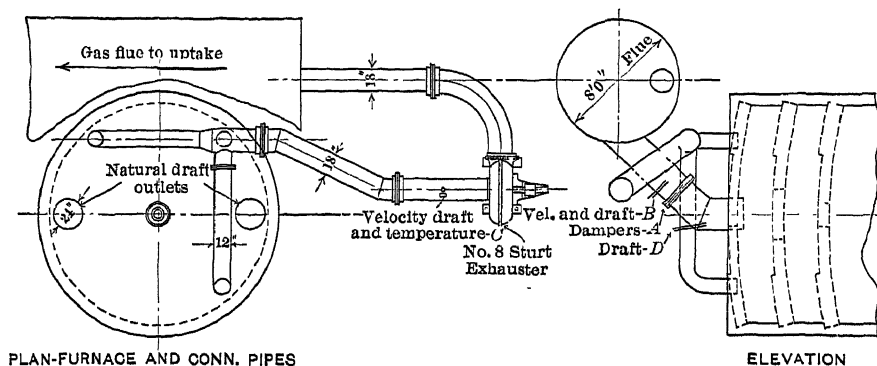
FIG. 1.—CONNECTIONS AT No. 2 MacDOUGALL FOR INDUCED DRAFT.

Summary of Data from Tests on No. 2 MacDougall Furnace. [a]

	Natural Draft.		Draft Induced by No. 8 Sturtevant Exhauster.		
Test Period, 1906.	Test No. 1	Test No. 2	Test No. 4	Test No. 5	Test No 6
	June 16–19.	June 20–22.	July 6–12.	July 13–28.	July 29– August 2.
Tons of concentrate per 24 hr., wet weight, . . .	30.382	42.885	39.803	43.014	45.858
Per cent. moisture in concentrate, ...	7.2	7.4	6.9	6.9	7.3
Tons of concentrate per 24 hr., dry weight, . . .	28.19	39.71	37.06	40.05	42.51
Per cent. sulphur in concentrate charged, ...	35.4	35.8	37.0	37.4	38.1
Per cent. sulphur in calcine produced, ...	5.3	11.6	8.8	9.6	11.5
Speed of exhauster, r. p. m., ...	. . .	. . .	1,450	1,454	1,503
Gas Discharged from Furnace : Average temperature, °F., ...	567	557	518	512	523
Average volume, cu. ft. per min. at observed temperature, ...	9,380	8,910	11,570	11,160	10,800

[a] Test No. 3 has been discarded.

From the above figures it will be seen that increasing the tonnage
of concentrate treated from 30.38 tons to 42.88 tons on natural draft
increased the percentage of sulphur in the calcine from 5.3 to 11.6,
whereas with induced draft (in Test No. 5) 43.01 tons of concentrate

were treated and calcine assaying 9.6 per cent. in sulphur was obtained. In other words, increasing the draft, and consequently the amount of air drawn through the furnace, effected a considerable increase in capacity. An idea of the amount by which the capacity was increased was obtained by comparing the work of MacDougall furnace No. 2 in Test No. 5 with the average work of all the other MacDougall furnaces during the same period. In making this comparison it should be noted that MacDougall furnace No. 2 was treating a feed of concentrate only, whereas the feed in the other furnaces, as was the regular practice at that time, was made up of about 92.6 per cent. of concentrate (wet weight), 4.8 per cent. of first-class ore screenings, 1.0 per cent. of tailing and 1.6 per cent. of dried slime. There would, therefore, probably be a slight advantage in favor of the test furnace.

Comparison of Test Furnace on Induced Draft with Average of all Other Furnaces.

Average.	No. 2 Furnace. (Induced Draft.)	Average of all Other Furnaces. (Natural Draft.)
Tons treated per 24 hr., wet weight, ...	43	33.8
Tons treated per 24 hr., dry weight, ...	40	31.5
Per cent. copper in calcine, ...	10.5	9.9
Per cent. sulphur in concentrate, ...	37.4	37.9
Per cent. sulphur in calcine, ...	9.6	9.0

From the above comparison, the following figures as to the percentage increase in capacity caused by the induced draft were obtained.

Relative Increase in Capacity of MacDougall Furnace No. 2 Operating Under Induced Draft in Comparison with Average of all Other MacDougall Furnaces.

Normal average charge per day, wet weight, ... 33.8 tons.
Average charge per day, wet weight, induced draft, ... 43.0 tons.
Increase in capacity, ... 27.0 per cent.
Gas discharged from furnace per minute, normal running (9,140 cu. ;ft.), 320 lb.
Gas discharged from furnace per minute, induced draft (11,150 cu. ft.), 410 lb.
Increase in amount of gas discharged, ... 22 per cent.

IV. EFFECT ON MACDOUGALL PLANT OF INCREASED DRAFT FROM NEW SMELTER STACK.

In June, 1909, the new stack and flue system having been finished, the MacDougall furnaces were started under the increased draft.

Sketches of the old and new flue system are shown in Figs. 2 and 3, respectively. The change in draft conditions for the MacDougall department is indicated by the following figures:

Draft in Inches of Water, MacDougall Furnace.

	Flue System.	
	Old.	New.
MacDougall dust chamber, ...	0.255	0.9
Flue necks from MacDougall furnaces to cross flue, ...	0.139	0.93

Using the same feed as under the old draft conditions, the first effect of the stronger draft was to put out all of the MacDougall furnaces. The reason for this was that, treating the same feed as formerly, the amount of heat generated by the oxidation of the sulphur remained practically unchanged; whereas, due to the increased draft, more air was drawn through the furnaces, more heat lost in the escaping gases, and consequently the furnaces became cold and went out. The first remedy tried was decreasing the draft by partly closing the dampers in the crosstake flue. After doing this the MacDougall department ran satisfactorily, but partly closing the dampers in the crosstake flue also decreased the draft for all the other smelter departments. After a few days, therefore, the dampers in the crosstake flue were again opened wide and the MacDougall draft decreased by partly closing the slides in the flue necks from the individual MacDougall furnaces. Also, the resistance to the passage of air through the furnaces was increased by covering one or two of the side drop holes on the second and fourth hearths with iron plates, reducing the drop hole area on these floors. As the foremen and furnacemen became familiar with operating the furnaces under the new conditions, the feed was gradually increased and more use made of the available draft. The increased capacity of the furnace under an increased draft, predicted by Mr. Klepinger's experimental work in 1906, was more than realized, as is shown by the following figures, taken from the monthly smelter reports. These figures show, for the MacDougall department, the tons treated per furnace day, tons treated per square foot of hearth area, and the average per cent. of sulphur in the calcine, for the 10 months immediately preceding and the 10 months immediately following the change to the new flue system.
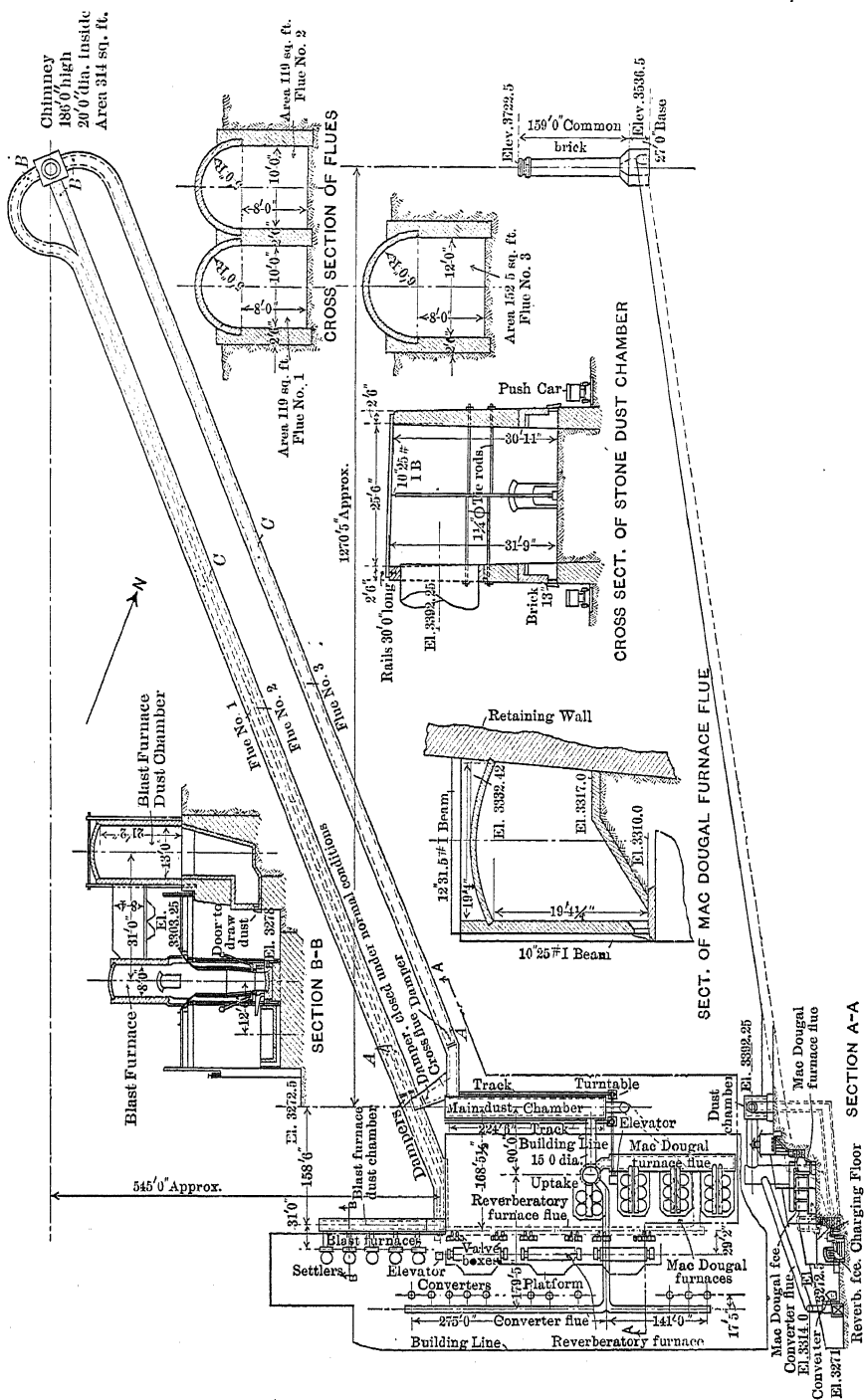
FIG. 2.—OLD FLUE SYSTEM.

FIG. 3.—NEW FLUE SYSTEM.

Data on MacDougall Department from Monthly Smelter Reports.

Month.	Concentrate.	Screenings.	Dried Slimes.	Flue Dust.	Limestone.	Total.	Total Tons per Square Foot of Hearth Area.	Per Cent. S in Calcine.
August, 1908,	28.7	...	...		...	28 7	0.032	9.1
September, 1908,	43.4	...	...	0.1	0.1	43.6	0.016	9.1
October, 1908,	41 4	...	0.1	0.9	...	42.4	0.045	8.9
November, 1908,	36.2	...	...	0.3	..	36.5	0.039	7.7
December, 1908,	37.8	...	(Copper Precip. 0 3)	0.1	. .	38.2	0.041	8.1
January, 1909,	33.1	.	...	...	...	33.1	0.036	8.8
February, 1909,	35.4		...	.	...	35.4	0.038	8.2
March, 1909,	35.1	...	0.1	.	...	35.2	0.038	8.4
April, 1909,	34.3	...	0.2	.	0.01	34.5	0.037	9.2
May, 1909,	3.33	...	.	.	...	33.3	0.036	9.1
Average,	35.9	...	0.2	0.1	...	36.1	0.039	8.7
July, 1909,	49.7	...	...	...	0.5	50.2	0.053	9.7
August, 1909,	40.8	...	0.7	...	0.4	41.9	0.046	8.6
September, 1909,	41.6	..	2.1	0.1	...	43.8	0.047	8.3
October, 1909,	40.2	...	2.4	...	0.3	42.9	0.046	8.2
November, 1909,	37.3	...	2.0	...	0.2	39.5	0.042	7.7
December, 1909,	Smelter down on account of railroad strike.							
January, 1910,	49.1	...	...	...	0.5	49.6	0.053	8.2
February, 1910,	51.9	...	...	...	1.7	53.6	0.057	8.7
March, 1910,	40.5	...	...	...	2.3	42.8	0.046	7.6
April, 1910,	44.8	1.2	...	..	1.2	47.2	0.050	7.9
Average,	44.0	0.1	0.8	..	0.8	45.7	0.049	8.3

V. INCREASING THE CAPACITY OF THE MacDOUGALL FURNACES.

Conditions Prior to Starting Experimental Work.

In the spring of 1910 efforts were begun to increase the tonnage treated per MacDougall furnace. From the figures tabulated above it will be seen that the new smelter stack and flue system had already brought about an increase in capacity from about 36.1 tons per furnace day to about 45.7 tons per furnace day, without raising the percentage of sulphur in the calcine. Treating tonnages higher than about 50 tons per furnace day, however, raised the percentage of sulphur in the calcine an undesirable amount and caused the furnaces to crust up heavily. Nevertheless it was hoped that by a systematic study of the conditions governing MacDougall work these difficulties could be overcome and a higher tonnage treated.

(A). Test of Arthur Crowfoot on Increasing Capacity.

(a). *Description of Test.*—The first step towards increasing the capacity of the furnaces was a test carried out under the direction of Arthur Crowfoot, Chief Sampler, during the period from May 3 to May 7,

inclusive, 1910. Four furnaces were selected for the test work, Nos. 1 and 3 operating as a unit and Nos. 18 and 19 operating as a unit. The essential differences between the two units were as follows :

1. The central drop hole connecting the third hearth with the fourth hearth in furnaces Nos. 1 and 3 measured approximately 20 in. from the shaft to the edge of the drop hole, while in furnaces Nos. 18 and 19 this opening measured approximately 14 in. The standard size for this third hearth drop hole had always been 14 in.; but a short time previous to starting this test the third hearth drop holes on furnaces Nos. 1, 2, 3, and 4 had been increased to 20 in., adopted from the practice at the Washoe Smelter at Anaconda.
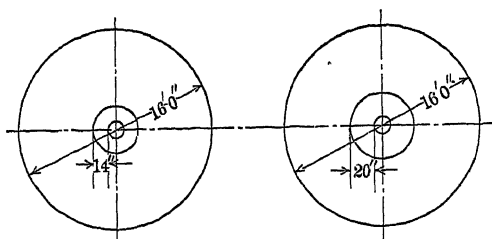
2. Speed of Rabble Arms. The rabble arms in furnaces Nos. 1 and 3 made one revolution in 50 sec., while the rabble arms in furnaces Nos. 18 and 19 made one revolution in 37 sec. The regular speed at which furnaces Nos. 18 and 19 ran, before starting the test, was one revolution in 53 sec., but this was increased to one revolution in 37 sec. about two weeks before starting the test.

The instructions for carrying out the tests were to start all furnaces on a fine concentrate feed of about 55 tons wet weight per 24 hr. per furnace, continuing this feed for 24 hr. on each furnace. After the first 24 hr., the rate of feed to each furnace was to be increased 5 tons per 24 hr. and this rate of feed kept up for 24 hr., and so on, increasing the rate of feed by 5-ton increments until reaching a rate of feed of 75 tons of wet concentrate per furnace per 24 hr.

The information obtained from the test was of considerable interest. A fractional part of the total data obtained is tabulated below.

Tonnage Test on MacDougall Furnaces.

(A). Average results obtained from Nos. 1 and 3 Furnace Unit.

Date of Test, 1910.	May 3.	May 4.	May 5.	May 6.	May 7.
Nominal tons wet conct to be fed per 24 hr., .	55	60	65	70	75
Total tons wet conct. fed per 24 hr., ...	52.72	59.05	64.54	69.17	59.18
Actual rate of feed, tons per 24 hr., ...	58.80	66.13	71.85	80.18	94.54
Hours furnace in operation, ...	21.51	21.30	21.71	20.70	15.79
Hours furnace down, ...	2.49	2.7	2.29	3.30	8.21
Per cent. of 24 hr. furnace down, ...	10.37	11.25	9.54	13.75	34.20
Draft reading on flue necks, inches water, . .	1.28	1.26	1.23	1.3	1.36
Concentrate Fed.					
Per cent. moisture, ...	7 5	8.0	6.7	7.6	7.1
Assay per cent. Cu, ...	8.80	7.55	8.20	8.4	8.45
Assay per cent. S, ...	36.0	32.8	34.6	34.4	35.8
Calcine Produced.					
Assay per cent. Cu, ...	10.55	9.75	10.35	10.1	10.25
Assay per cent. S, ...	8.2	9.2	11.3	12.2	12.6

(B). Average results obtained from Nos. 18 and 19 Furnace Unit.

Nominal tons wet conct. to be fed per 24 hr.,	55	60	65	70	75
Total tons wet conct. fed per 24 hr., ...	52.45	58.31	60.91	59.41	63.01
Actual rate of feed, tons per 24 hr., ...	64.20	72.72	71.40	74.16	79.84
Hours furnace in operation, ...	19.58	19.24	20.49	19 25	18.92
Hours furnace down, ...	4.42	4.76	3.51	4.75	5.08
Per cent. of 24 hr. furnace down, ...	18.42	19.83	14.60	19.79	21.17
Draft reading on flue necks, inches water, . .	1.16	1.09	1.09	1.12	1.16
Concentrate Fed.					
Per cent. moisture, ...	8.4	6.6	6.6	7.1	7.1
Assay per cent. Cu, ...	6.40	8.10	8.25	8.65	8.0
Assay per cent. S, ...	32.8	33.4	34.8	35.0	35.4
Calcine Produced.					
Assay per cent. Cu, ...	10.35	9.65	9.85	9.95	10.1
Assay per cent. S, ...	8.4	7.8	8 8	9.6	10.4

(b). *Discussion of Results Obtained from Test.*—The most interesting fact clearly brought out by this test was that increasing the feed to the furnaces did not increase the percentage of sulphur in the calcine as much as expected. In fact, the 18-19 furnace unit with the smaller third hearth drop hole and greater speed of revolution of the central shaft produced very satisfactory calcine. Previously it had been thought that, under a given draft, the number of pounds of sulphur per 24 hr. which a furnace would eliminate was, very approximately, a constant, and that on increasing the feed, and consequently the pounds of sulphur per 24 hr. delivered to a furnace, the excess of sulphur fed would nearly all go into the calcine. The test showed that on increasing the feed of concentrate more pounds of sulphur were

burned per minute and per square foot of hearth area, and the furnaces ran much hotter. In fact, the main difficulties experienced were on account of too high a temperature being developed. The unusually large amount of heat generated caused the drop holes to become built up, and heavy accretions to form on the roofs. The conclusions drawn from the test were that a high tonnage feed of straight concentrate, under the conditions existing at that time, caused the furnaces to become too hot for practical operation.

(1). Third Hearth Center Drop Hole.—Below are tabulated data on the amount of time lost by the two furnace units:

Nominal Tons Treated.	Percentage of Time Down in 24 Hr.		
	Furnaces 1 and 3.	Furnaces 18 and 19.	Differences.
55, ...	10.37	18.42	8.05
60, ...	11.25	19.83	8.58
65, ...	9.54	14.60	5.06
70, ...	13.75	19.79	6 04
75, ...	34.20a	21.17	....

a About 23 per cent. due to mechanical breakdown of furnace No. 3.

From the above tabulation it will be seen that the 18–19 furnace unit with the 14-in. third hearth central drop lost on an average about 7 per cent. more time than the 1–3 furnace unit with the larger third hearth central drop hole. The reason for this was that the smaller third hearth drop hole, on account of the higher velocity of the gases and the concentration of heat at this point, caused heavy accretions to build up on the roof above this drop hole. Practically all of the additional time lost by the 18–19 furnace unit over the 1–3 furnace unit was due to barring accretions from the roof of the third hearth.

(2). Speed of Rabble Arms.—The observations showed that in treating high tonnages, increased speed was advantageous, for the reason that it kept shallower beds on the furnace hearths, and there was less banking up of material in front of the rabble arms.

General Discussion of Data Obtained from Above Test.—The data obtained from this test outlined the way for future work. It was necessary to have the furnaces run cooler, which could be accomplished by either drawing more air through the furnaces and consequently discharging more heat in the waste gases, or else by decreasing the sulphur content (fuel value) of the feed. Also, for treating high tonnages, the test indicated that the larger size of drop hole and increased rate of speed were in all probability desirable.

(B). Investigation Carried Out by John A. Church, Jr.

Immediately following the above test, John A. Church, Jr., was detailed to make a special investigation looking towards improving the work of the MacDougall department and increasing the tonnage treated per furnace. At this point it should be noted that credit for the excellent results which were obtained from this investigation is due both to Mr. Church and to George S. Crouse, calcine foreman. Mr. Crouse has been employed in the Calcine Department since December, 1893, working on the Brueckner cylinders before the first installation of MacDougall furnaces; and has been foreman since 1898. He has always gladly co-operated with the Testing Department in its MacDougall work. The writers in particular are indebted to Mr. Crouse, who assisted them in their later work and generously placed at their disposal the information obtained from his long experience and keen interest in MacDougall work.

Mr. Church began his investigation by observations on MacDougall furnace No. 19. Acting on the information obtained from Mr. Crowfoot's test, this furnace was operated on a high tonnage, but the fuel value of the feed was lowered by mixing screenings and lime rock with the concentrate. The feed treated was made up of 20 tons of concentrate and 6 tons of foreign material per shift. By "foreign material" is meant any constituent of the feed lower in fuel value than the concentrate, such as slime or first-class ore screenings. Shifts are all 8 hr. The furnace had a small third hearth drop hole, measuring 14 in. from the shaft to the edge of the drop hole, and ran at high speed, the rabble arms making one revolution in 38 sec. The draft was regulated by dampers in the flue necks, and by closing all doors except one on the sixth hearth. The furnace handled the 20 tons of concentrate per shift satisfactorily and the foreign material mixed with the concentrate kept the temperature of the furnace from becoming too high. It was found, however, that the work of keeping the roof clean above the third hearth drop hole was excessive for normal running.

On May 17, therefore, MacDougall furnace No. 18 was equipped with an enlarged third hearth drop hole and started for parallel comparison with furnace No. 19. The conditions under which the two furnaces ran were as shown below:

Conditions.	Furnace No. 18	Furnace No. 19.
Time of one revolution, ...	38 sec.	38 sec.
Size of third hearth drop hole (shaft to edge of drop hole), ...	20 in.	14 in.
Draft, inches water, ...	0.8 to 0.9	0.8 to 0.9
Tonnage of concentrate, ...	20 per shift.	20 per shift.
Tonnage of foreign material, . . .	As much as furnace would take.	7–8 tons per shift.

Both furnaces had two of the six drop holes covered on the second hearth, and two closed on the fourth hearth. Two doors were left open on the sixth hearth.

A few days of operation under the above conditions showed that furnace No. 18 with the larger third hearth drop hole ran considerably cooler and crusted up less than furnace No. 19. Also furnace No. 18 would not take as large a tonnage of foreign material without becoming too cold. Although by means of feeding foreign material along with the concentrate the temperature of No. 19 furnace as a whole could be maintained at about the right point, on account of the concentration of heat at the third hearth drop hole and the high velocity of the gases at this point, the work of keeping the roof above the third hearth clean still continued to be excessive. This test therefore corroborated the earlier work of Mr. Crowfoot, and demonstrated for high tonnages the advantages of the larger third hearth drop hole. Accordingly, when on May 24 a leak in the second hearth made it necessary to shut down furnace No. 19 for repairs, advantage was taken of the shut down to enlarge the third hearth drop hole to 20 in.

While enlarging the third hearth drop hole of a furnace greatly reduces the work of keeping the furnace clean, it also has the undesirable effect of making the calcine run higher in sulphur. A year or two later a means was found of retaining the smaller drop hole, and yet avoiding excessive incrusting on the third hearth roof. Two extra drop holes were cut in the third hearth close to the edge of the center drop hole. The inner rakes on the third hearth rabble arms were set so as to push the material away from the center, and therefore all material passing from the third to the fourth hearths was forced to pass through the two extra drop holes. This meant dropping a large bulk of material four times in every revolution of the central shaft, instead of having the material showering down continuously over the edge of the center drop hole. It reduced the amount of incrusting on the third hearth roof and localized what incrustations formed over the extra drop holes, making the work much easier for the furnacemen. The 14-in. center drop hole combined with the two extra drop holes is the present practice for the third hearth. It is mentioned here because, although during the work in the spring of 1910 to increase the tonnage per furnace it was found that the 20-in. was an improvement over the 14-in. third hearth drop hole, later work as noted above has made it desirable to again change back to the smaller drop hole.

Returning to Mr. Church's investigation, while furnace No. 19 was

down for repairs and to have the third hearth drop hole enlarged, observations were kept up on furnace No. 18.   More doors were gradually opened on the sixth hearth until finally the furnace was run with all four of the sixth hearth doors open.   (The doors on the upper hearths are always kept closed except when they must be opened for cleaning, or pulling lumps out of the furnace.)   The opening of each successive door brought about an improvement in the condition of the furnace, showing that previously not enough oxygen had been supplied.   Similarly, the dampers in the flue necks were opened wider, improving the work of the furnace, until in time the full available draft of the lateral flue was being used, about 1.2 in. of water.

The chief difficulties experienced with No. 18 were due to the sensitiveness of the furnace.   Changes in the quality of the feed, irregular feeding of foreign material, and other adverse conditions would often cool the furnace too far, necessitating the use of the oil machine to restore a normal working temperature.   The first improvement in maintaining the furnace at a steady temperature was made by closing one additional second hearth drop hole.   This made three second hearth drop holes closed in all.   At first it was thought that the increased amount of heat obtained by decreasing the second hearth drop hole area was due to increasing the frictional resistance to the passage of gas through the furnace, allowing less air to enter the furnace and less heat to be carried away in the waste gases.   The improvement which had been made in the work of the furnace, however, by opening more doors on the sixth hearth and opening wide the dampers in the flue necks, showed that this was not the case.   The drop hole area on the second hearth, even with three drop holes covered, was large enough not to throttle to any appreciable extent the gases traveling through the furnace.   The effect, therefore, of closing one extra drop hole was to concentrate the ascending hot gases, which had previously been passing through four drop holes, into three, thus intensifying the heat at these three points.   Near the outer edge of the second hearth is where the concentrate should first begin to burn, and a certain temperature is required for its ignition.   As furnace No. 18 had a tendency to run cold, closing an extra second hearth drop hole, therefore, benefited the furnace by helping to secure proper ignition for the incoming concentrate.   Decreasing the second hearth drop hole area probably also helped by lessening the amount of heat lost by radiation from the third hearth.

Another improvement made in maintaining furnace No. 18 at a good working temperature was increasing the amount of concentrate fed. The feed of concentrate was raised first to 22 and later to 24 tons per

shift. With the higher tonnage the furnace developed more heat and it was found advisable to remove one of the second hearth drop hole covers, again returning to the condition of four open and two closed. On June 2, MacDougall furnace No. 19 was again started, the third hearth drop hole having been enlarged. The furnace was operated in every respect similar to No. 18. It was found that for some unexplained reason No. 19 normally ran hotter than No. 18, and as the tonnage of concentrate fed was increased to 24 tons per shift it was found that No. 19 ran best with all of the second hearth drop holes uncovered. No. 18 throughout the testing work was found to give the best results with two of these drop holes covered. It will be seen, therefore, that the proper second hearth drop hole area is a variable quantity, depending on the temperature conditions of a furnace, quantity of feed treated, and so on.

On June 17 it was decided to run furnace No. 22 against furnaces Nos. 18 and 19 to determine the question of the relative advantages of high and low speed, all other conditions being the same. Accordingly a course of brick was knocked out around the third hearth drop hole of No. 22, and instructions were given to gradually raise the tonnage of concentrate to 24 tons per shift. Nos. 18 and 19 continued to revolve once in 38 sec. and No. 22 was left unchanged at one revolution in 53 sec. The operation of all three furnaces during about a 10-day period showed that the high-speed furnaces gave less trouble from material banking up in front of the arms and in general produced calcine assaying lower in sulphur. A discussion of the relative advantages of high-speed, condensed from the report of Mr. Church, is presented herewith.

(a). *Advantages of High Speed.*—1. Effect of Increased Speed on Depth of Bed.—The experimental work showed that for high tonnages the increased speed gave better results, in that the depth of bed on the hearths was kept lower and there was less tendency for the feed to bank up in front of the arms. The reason for the shallower bed in a high-speed furnace is as follows: Assume that in a MacDougall furnace the depth of bed remains constant, then every revolution of the central shaft moves a constant amount of material at any one point on the hearth over a radial distance equal to the width of two furrows. If the speed is increased and the depth of bed remains constant, every revolution moves the same amount of material the same distance as before, but in a given time the greater number of revolutions will move a greater tonnage. If, however, the speed is increased and the tonnage remains constant, every revolution will move a less quantity of material than before, that is, the depth of bed will be reduced.

2. *Effect of Increased Speed on Degree of Calcination.*—Increasing the speed of a MacDougall furnace, means not only a decrease in the depth of bed with constant tonnage, as shown above, but also means that the bed is stirred more frequently.   Every passage of a rabble arm, by stirring the bed, presents a freshly exposed surface to the action of the oxygen-bearing gases.   The best speed is that which exposes a fresh surface just as rapid oxidation of the sulphur is beginning to die in the old surface.   The time required for this rapid oxidation of the surface to cease varies with the character of the material, the amount of oxygen in the gases, the temperature of the bed, temperature of the gases, and many other factors.   In general, however, it may be said that the period during which the surface of the bed under goes rapid oxidation is shorter for the fifth and sixth hearths than for the third and fourth hearths.   Up to the point at which a surface is buried while still undergoing rapid oxidation of its sulphur, increasing the speed tends to increase the rate of elimination of sulphur.

On the third floor, the speed of one revolution in 38 sec. undoubtedly made the stirring too rapid.   On the fourth floor observations indicated that the increased speed was probably about right, while for the fifth and sixth hearths a still higher speed might have been used to advantage.

(C). Effect on Regular Operation of Department by Applying Results Obtained from Experimental Work.

By July 1, 1910, the advantages of increased speed had been demonstrated to the extent that it was deemed advisable to change all of the furnaces in Battery IV. to the speed of one revolution in 38 sec.; later, all the furnaces in the MacDougall department were increased in speed, the speeds in use at present (May, 1913) being as shown below:

Former Speed of Revolution of Shaft.   *Present Speed of Revolution of Shaft.*

One revolution in 53 sec.   Battery I. (4 furnaces), one revolution in 45 sec.

Batteries II., III., and IV. (18 furnaces), one revolution in 38 sec.

At the same time that all of the furnaces in Battery IV. were changed to the increased speed (July, 1910), the furnaces in operation in that battery were also brought up to the increased tonnage of 24 tons of concentrate per shift.   Later, all of the MacDougall furnaces were raised to the new tonnages.   The effect on the MacDougall de-

partment, as the improved methods of operation developed from the experimental work were gradually applied to the regular daily work of the furnaces, is shown by the figures tabulated below. These figures are taken from the Monthly Smelter Reports.

MacDougall Department.

Average Tonnage Treated per Furnace Day.

Month.	Concentrate.	First-Class Ore Screenings.	Miscellaneous.	Total Cupreous Material.	Limestone.	Total.	Total Tons per Square Foot of Hearth Area.	Per Cent. S in Calcines.
April, 1910...	44.8	1 2	....	46.0	1.2 ˙	47.2	0.050	7.9
May, 1910...	46.9	3.9	0.1	50.9	2.6	53.5	0.057	7.6
June, 1910 .	47.2	3.6	0.6	51.4	3.6	55.0	0.059	8.0
July, 1910...	55.2	6.7	0.1	62.0	4.6	66.6	0.071	9.0
August, 1910 .	52.4	4.0	....	56.4	3.9	60.3	0.064	8.2
September, 1910 .	67.5	3.9	....	71.4	6.5	77.9	0.083	9.8
October, 1910 .	68 1	5.3	....	73.4	8.2	81.6	0.088	8.6
November, 1910...	73.8	4.6	....	78.4	9.8	88.2	0.094	8.0
December, 1910 .	73.1	4.2	....	77.3	11.0	88.3	0 094	8.4
January, 1911 .	67.6	2 4	....	70.0	11.9	81.9	0.088	8.4
February, 1911.	70.5	4.0	....	74.5	14.3	88.8	0.095	10.6
March, 1911...	75.1	0.3	....	75.4	10.3	85.7	0.092	11.4

(D). Fundamental Principles Involved in Increasing the Tonnage.

Before the experimental work to increase the capacity of the Mac-Dougall furnaces was started, the reasons for not treating more than about 14 tons of concentrate per furnace per shift were that increasing the tonnage raised the percentage of sulphur in the calcine, caused the furnaces to become too hot, especially locally on the roof above the 14-in. third hearth drop hole, and made the furnaces crust up heavily, necessitating an excessive amount of cleaning. The problem of treating increased tonnages therefore involved the following factors :

(*a*). Supplying sufficient oxygen and stirring the hearths frequently enough to burn the increased amount of sulphur.

(*b*). Regulating the heat from the increased amount of sulphur (fuel) burned per square foot of hearth area so that the furnaces would not become too hot.

(*c*). Regulating the drop hole area so as to avoid too great a concentration of heat and too high a velocity of the gases through the drop holes, thus preventing the furnaces from building up too heavy incrustations, particularly on the roof above the third hearth center drop hole.

The specific changes made, through which the treating of higher tonnages became possible, were as follows :

(*a*). The furnaces were operated under the full available draft (about

1.1 in. of water), with no dampers in the flue necks and with four doors open on the sixth hearth, so as to draw enough air through the furnace to supply the necessary amount of oxygen.

(*b*). The speed of revolution of the central shafts was increased from one revolution in 53 sec. to one revolution in 45 sec. on the first battery (four furnaces), and to one revolution in 38 sec. on all the other batteries (18 furnaces). This increase of speed had two effects:

1. It permitted an increase in tonnage without any particular increase in the depth of bed on the hearths, thus helping the furnaces mechanically.

2. It stirred the beds more frequently and aided in the more rapid oxidation of sulphur by bringing the sulphur in contact with sufficient oxygen. In a given time, to obtain the same degree of calcination, a furnace treating a higher tonnage has to eliminate a larger quantity of sulphur.

(*c*). First-class ore screenings and lime rock were mixed and fed with the increased tonnage of concentrate to reduce and regulate the temperature of the furnaces.

(*d*). The area of the third hearth center drop hole was increased. This reduced the amount of heat which had formerly been concentrated at this point and decreased the velocity of gases through this opening, resulting in permitting the use of a higher tonnage without an excessive amount of crusting on the third hearth roof.

(*e*). The second hearth drop hole area was regulated so as to obtain sufficient heat at the edge of the second hearth to ignite the incoming concentrate.

VI. Treatment of an Increased Amount of Ore Screenings.

The experimental work described above had resulted in increasing the efficiency of the MacDougall department by increasing the capacity of the individual furnaces. That is, the total output of the department remained about the same, but the work was done by a fewer number of furnaces, resulting, as will be shown later, in a considerable reduction in labor cost. A short time later it also became desirable to increase the total output of the MacDougall plant to supply the feed for an additional reverberatory furnace. The MacDougall furnaces were already handling practically the entire output of fine concentrate from the concentrator, and no particular increase in the supply of fine concentrate could be looked for. To increase the amount of material produced by the MacDougall department, therefore, it was desired to treat a larger amount of first-class ore screenings, of which a sufficient supply could be obtained.

In the previous work, the ore screenings, being lower in fuel value than the concentrate, had been used as a cooling agent to keep down the temperature of the furnaces. The problem now became, instead of running the MacDougalls so as to treat the best possible tonnage of concentrate, using screenings only as a cooling agent, one of running the MacDougalls so as to treat the best possible tonnage of screenings by utilizing the available heat in the most efficient way. The means used for treating a large tonnage of screenings were as follows:

1. Operation of the furnaces on a high tonnage of concentrate.

2. Proper regulation of the second hearth drop hole area.

3. Withdrawal of lime rock from the MacDougall feed, its place being taken by screenings.

1. Operating Furnaces on a High Tonnage of Concentrate.—Screenings being low in fuel value and having a cooling effect, as previously stated, it was necessary to keep the furnace as hot as possible. Increasing the feed of concentrate meant burning more pounds of sulphur per square foot of hearth area per minute and developing more heat. The concentrate feed to each individual furnace was therefore maintained as high as possible without unduly raising the percentage of sulphur in the calcine.

2. Proper Regulation of Second Hearth Drop Hole Area.—In order to secure proper ignition for the feed, the second hearth drop hole area was regulated so as to obtain as high a temperature at the outer edge of the second hearth as possible without too much throttling of the draft.

3. Replacement of Lime Rock by Screenings.—Lime rock having no fuel value, it was found that its place could be taken by a greater quantity of screenings, the screenings carrying an average of from 18 to 20 per cent. of sulphur and therefore having a fuel value. The construction of the MacDougall furnaces was therefore changed so that on two furnaces in each battery lime rock could be fed directly to the outer edge of the sixth hearths. All the lime rock treated by the MacDougall plant was fed to the sixth hearths of these furnaces.

In addition to the possibility of treating a larger tonnage of screenings, there were a number of other advantages in favor of this arrangement. One advantage was that the amount of lime rock mixed in with the calcine was not limited by the temperature conditions of the furnaces, for feeding lime rock directly to the edge of the sixth hearth had very little cooling effect. Another advantage was that the lime rock being greater in size than any other constituent of the MacDougall feed, its removal from the main body of the furnace lessened the danger of breaking rakes by the wedging of large fragments under the

blades. Furthermore, there had been no particular advantage in sending the lime rock completely through the furnaces. It contained no sulphur to be eliminated, and, as the temperature in the furnaces was not great enough to burn lime rock to lime, it underwent no chemical changes. By sending it directly to the outer edge of the sixth hearths it was warmed as much as if it had passed entirely through the furnaces and was nearly as well mixed with the calcine.

VII. MAXIMUM PERCENTAGE OF ORE SCREENINGS IN A MACDOUGALL FEED.

At this point, the experimental work on increasing the efficiency of the MacDougall department was taken up by Mr. Corwin, who carried out the tests to determine the maximum percentage of screenings which could be treated in a MacDougall feed, and to further increase the tonnage of concentrate treated per furnace. Tests on the Repath-Marcy furnace, described later, were made by both of the writers, and the experimental work on the Crouse furnace was taken up by Mr. Rodgers. All of the testing was done under the direction of Mr. Krejci and Mr. Crowfoot, and in co-operation with George S. Crouse, MacDougall foreman, to whom a great amount of credit for the steady improvement in MacDougall work is due.

The test to determine the maximum percentage of screenings, even up to 100 per cent., which could be treated in a MacDougall feed was started in December, 1910. The results of this test are of unusual interest as showing the wide range in sulphur content of the material which can be successfully roasted in a MacDougall furnace.

At the time of starting the test about 8,820 lb. of screenings could be treated per furnace per shift, mixed with about 50,000 lb. of concentrate. The ratio was therefore about 25 tons of concentrate (wet weight) to 4.41 tons of screenings per shift, or 85 per cent. of concentrate to 15 per cent. of screenings. The screenings were fed intermittently during a shift, and, as they always had a cooling effect, they were used only when the furnaces were running particularly hot. This led the furnacemen to believe that the continuous treatment of a 100 per cent. screenings feed would be impossible, and in carrying out the test it was necessary to increase the percentage of screenings by slow degrees. The successful solution of the problem depended on keeping the furnace sufficiently hot.

MacDougall furnace No. 16 was used for the test work, and whenever necessary the temperature of the furnace was raised by means of either or both of the two following:

1. Compressed air.

2. Coal and oil.

1. Compressed Air.—The advantage of compressed air in keeping a MacDougall furnace hot was discovered by noticing the effect produced by air blown into the furnace through an oil machine after the oil in the machine had been exhausted. It was observed that the jet of air on the fourth hearth caused more rapid oxidation of the sulphur, developing more heat and increasing the temperature of the furnace. The increase in rapidity of the oxidation was caused by the air impinging on the hearth, bringing the calcine into intimate contact with fresh oxygen.

In the test to determine the maximum percentage of ore screenings which could be used in the feed to a MacDougall furnace, furnace No. 16 was first equipped for compressed air with three 0.25-in. pipes inserted through the observation holes of the fourth hearth doors. It was found that air blown through these pipes aided greatly in increasing the temperature of the furnace, and many times when the furnace showed signs of becoming cold a recovery was effected by means of compressed air without the use of oil or coal. One disadvantage, however, was that there was an increased amount of incrusting on the rabble arms, due to sparks produced by the air blowing on the bed. A large amount of this incrusting was done away with, and better results obtained from the air, by replacing the 0.25-in. pipes by 0.75-in. pipes. This change made it possible to slightly decrease the velocity of the air entering the furnace, at the same time increasing the volume. Thus the amount of sparking was reduced and the hearth area affected by the air increased.

A further improvement was made in the method of introducing the compressed air into the furnace before the completion of the test. On the fourth hearth, three 0.75-in. pipes each 4.5 ft. long were inserted through holes drilled in the shell and brickwork of the furnace. These pipes were run towards the center of the furnace along the fourth hearth roof, leaving just enough room for the rabble arms to pass beneath them. Just outside of the shell of the furnace the pipes were each connected to a rubber hose delivering compressed air from the converter air main. Inside of the furnace the pipes each had two series of 0.25-in. holes drilled in them, extending from the brick wall of the furnace to the ends of the pipes. The holes were drilled 1.5 in. apart, and the two series were each 0.5 in. apart, directing the air at nearly right angles down on the bed. These pipes caused the compressed air to affect a large area of the bed, and because the air was directed against the bed at nearly right angles,

sparking was practically eliminated. Pipes were also used to intro-
duce air to the fifth hearth, the arrangement being exactly the same
as for the fourth. It should be remarked here that, although the
equipment described above was found highly satisfactory during the
short period of the test, it would not be suitable as a permanent in-
stallation, for the reason that the pipes are liable to become bent by
the heat and interfere with the arms, and also because they make it
difficult to clean the roofs of the hearths where they are placed.

The experiment was also tried of blowing air in on the third hearth,
but the results were unsatisfactory. The temperature of the third
hearth is too variable, and unless the air is blown against a hot bed it
does not increase the rate of oxidation of the sulphur, having simply
a cooling effect.

2. Coal and Oil.—As the percentage of ore screenings in the feed
to MacDougall furnace No. 16 was increased, the furnace could not
be kept hot enough by the use of air only, and it became necessary
to use a small amount of slack coal. This coal was fed through a
different hopper from the screenings and concentrate, a fairly even
feed being obtained by poking the coal through the hopper a little at
a time and allowing it to mix with the screenings and concentrate on
the first hearth. Whenever the furnace received a slight set back
oil was also used to keep it hot, the oil being blown in on the fourth
hearth in the usual manner.

On night shift of Feb. 7, 1911, a 100 per cent. ore screenings feed
was first treated in the furnace and from then until Feb. 12, the fur-
nace was maintained on this feed. Coal and air were used continu-
ously to keep the furnace hot, and oil when necessary. To show the
low fuel value of ore screenings as compared with concentrate,
the assays on monthly samples of screenings and fine concentrate
treated in the MacDougall department during January, 1911, are pre-
sented below :

Assay Results on Monthly Samples, January, 1911.

To MacDougalls.	Electro-lytic Cu.	SiO₂.	FeO.	Al₂O₃	CaO.	S.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
First-class ore screenings, . . .	7.10	46.9	16.6	9.7	0.1	17.4
Fine concentrate, ...	7.45	24.0	32.8	6.1	0.1	33.4

A record of MacDougall furnace No. 16 for the last 19 days of the
test is given herewith.

Record of MacDougall Furnace No. 16 During Test on Treatment of Ore Screenings.

Date, 1911.	Tons of Material Charged.			Per Cent. of Feed.		Fuel Used.		Fuel per Ton of Feed.		S in Calcines.
	Concentrates.	Screenings.	Total.	Concentrates.	Screenings.	Coal.	Oil.	Coal.	Oil.	
						Pounds.	Gallons	Pounds.	Gallons	PerCnt. No sample
January 25 ..	65 16	17 75	82 91	78.59	21 41	......	.....	45.26	0.83	8.9
January 26	45 71	26.97	75 68	64.36	35 64	3,425	63	45.41	0.96	12 8
January 27 ...	45 68	31 40	77.08	59.26	40.74	3,500	74	59 37	0 62	7.2
January 28 .	34.64	34.42	69 06	50.16	49.84	4,100	43	45.78	0.52	5.6
January 29.. ..	30.51	36 11	66 62	45.80	54.20	3,050	35	31.39	0.54	6.0
January 30...	24 87	40 15	64.52	37.77	62.23	2,025	35	47.87	0 58	5.7
January 31 ...	24.39	46.64	71.03	34.34	65.66	3,400	41	65 40	0.84	6.7
February 1	18.80	48.21	66 51	27 51	72.49	4,350	56	130 67	0.21	6.6
February 2...	14.21	66.95	81.16	17.51	82.49	10,605	17			
February 3.. February 4. February 5. February 6.	No screenings used from Feb. 3 to 6, inclusive.									
February 7....	7.80	56.32	64.12	12.16	87.84	9,255	24	144.34	0.37	7.3
February 8	. .....	74 13	74.13	.. ..	100	13,745	8	185.42	0.11	8.0
February 9	....	74.35	74 35	.. ..	100	11,430	46	153.73	0.62	6.2
February 10...	. ...	69.80	69.80	.. ..	100	10,260	47	146.99	0 67	6.5
February 11..	....	65 80	65.80	....	100	8,335	24	126.67	0.36	5.6
February 12....	.. ... .	66.52	66 52	....	100	7,955	5	119.59	0.07	4.9

VIII. TESTS TO FURTHER INCREASE CAPACITY OF FURNACES.

Immediately on the completion of the test to determine the maximum percentage of ore screenings which could be treated in a MacDougall feed, experimental work was again resumed on increasing the tonnages. MacDougall furnace No. 16, which had just been used on the screenings test, was used also for the tonnage test. The object in view was to treat in this furnace the highest tonnage consistent with good calcination. As already stated, this furnace was equipped with pipes for blowing air on the beds of the fourth and fifth hearths. In the screenings test the air had been used to increase the rate of oxidation, in order to increase the temperature of the furnace. In the high tonnage test compressed air was used to increase the rate of oxidation, but not to make the furnace run hotter. With the high tonnage of concentrate, and therefore the large amount of sulphur burned per square foot of hearth area, the furnace ran so hot that it was always necessary to feed screenings along with the concentrate to cool the furnace and prevent the material on the third hearth from fusing. The object of increasing the rate of oxidation was not, therefore, to raise the temperature of the furnace, but to improve the degree of calcination. The speed of revolution of the central shaft was not changed, remaining at one revolution in 38 sec.

The high-tonnage test was carried out during the period from Feb. 13 to Mar. 3, inclusive, 1911. The maximum tonnage treated in any

one 24-hr. period, 105.08 tons, was put through the furnace on Feb. 23. Below is given a record of the furnace for the three shifts of Feb. 23:

Record of MacDougall Furnace No. 16, Feb. 23, 1913.

Shift.	Tons Material Charged.		Total Feed. Tons.	Per Cent. of Feed.		Fuel		Per Cent. S in Calcine.
	Concentrate.	Screenings.		Concentrate.	Screenings.	Coal. Pounds.	Oil. Gallons.	
Day, ...	32.48	1.60	34.08	95.31	4.69	. . . . .	. . .	12 2
Afternoon, ...	32.14	3.16	35.30	91.05	8.95	. . .	. . . .	13.1
Night, ...	32.47	3.23	35.70	90.95	9.05	. .	. . . .	11.6
Total 24 hr., ...	97.09	7.99	105.08	. . .	. . .	. . . .	. . . .	. . .

Although it was found that on some days tonnages of around 100 tons per 24 hr. could be treated, producing calcine assaying about 11.5 per cent. in sulphur, the test showed that these tonnages could not be treated continuously on account of trouble with the driving machinery. Sometimes the feed would become so heavy on the first hearth that the machinery would refuse to carry it, and it would be necessary to rabble the material by hand on to the second hearth. Breakages of the driving machinery were of frequent occurrence, necessitating shutdowns for repairs. During these shutdowns the furnace would lose not only the tonnage which would have been treated had it been possible to continue the running of the furnace, but also, on account of the cooling off of the furnace during the shutdown, it would lose tonnage on account of the necessity of reducing the feed on starting up, until again reaching a normal working temperature.

The heavy load on the driving machinery was the result of increasing the tonnage treated without increasing the speed of revolution of the central shaft. As explained earlier in this report, increased tonnage with constant speed means deeper beds on all the hearths. The test showed that with the central shaft making one revolution in 38 sec. 100 tons per 24 hr. could not be treated continuously in the furnace, on account of overloading the driving machinery. Increasing the speed of the central shaft was not tried and might or might not have caused too frequent stirring of the bed. Whether or not this tonnage could have been treated successfully by means of increased speed is still an open question.

IX. TESTS TO DECREASE PERCENTAGE OF FLUE DUST MADE BY
MACDOUGALL FURNACES.

The month following the completion of the high-tonnage test, March, 1911, experimental work was undertaken to try to reduce the percentage of the flue dust made by the MacDougall furnaces. The flue dust produced by a MacDougall furnace is caused by the material dropping from hearth to hearth through the rising current of gases. With a strong draft small particles of partly roasted concentrate are picked up by the gases and carried along into the flue. Similarly, on the lower floors the incrustations around drop holes are the result of the material falling through the rising gases. Small particles of calcine are picked up by the draft, and as during the fall through the gases they undergo very rapid oxidation and become partly fused, when they are thrown up against the roof they stick there, forming accretions. The experimental work to reduce the amount of flue dust was therefore based on eliminating the fall of material through the upward current of gases.

(A). Repath-Marcy Furnace.

The first tests were made with the method invented and patented by Repath and Marcy. This method provides drop holes for the passage of the material from hearth to hearth, which are sealed by the material itself against the gases, and also provides separate passageways from hearth to hearth for the gases. A furnace fulfilling these requirements was designed by the engineers at the Great Falls Smelter and MacDougall Furnace No. 17 was remodeled to conform with the design. This furnace will be referred to as the Repath-Marcy furnace. The construction of the furnace is shown in the accompanying sketch, Fig. 4.

The Repath-Marcy furnace was started on Mar. 15, 1911. At first there were a number of mechanical difficulties encountered which interfered with the regular discharge of the material from hearth to hearth. At times the drop holes would allow the material to run through too fast and would become open for the passage of gases. At other times they would not allow the material to pass through fast enough and would become choked. Another difficulty met with was the excessive loss of heat by radiation from the gas passageways. ˙ The furnace was therefore shut down after a few days' running, the defects in the drop hole construction were remedied, and the gas passageways lined with brick. On starting up again after making these changes, with the furnace running satisfactorily mechanically and as nearly as possible in accordance with the ideas of the designers, it was
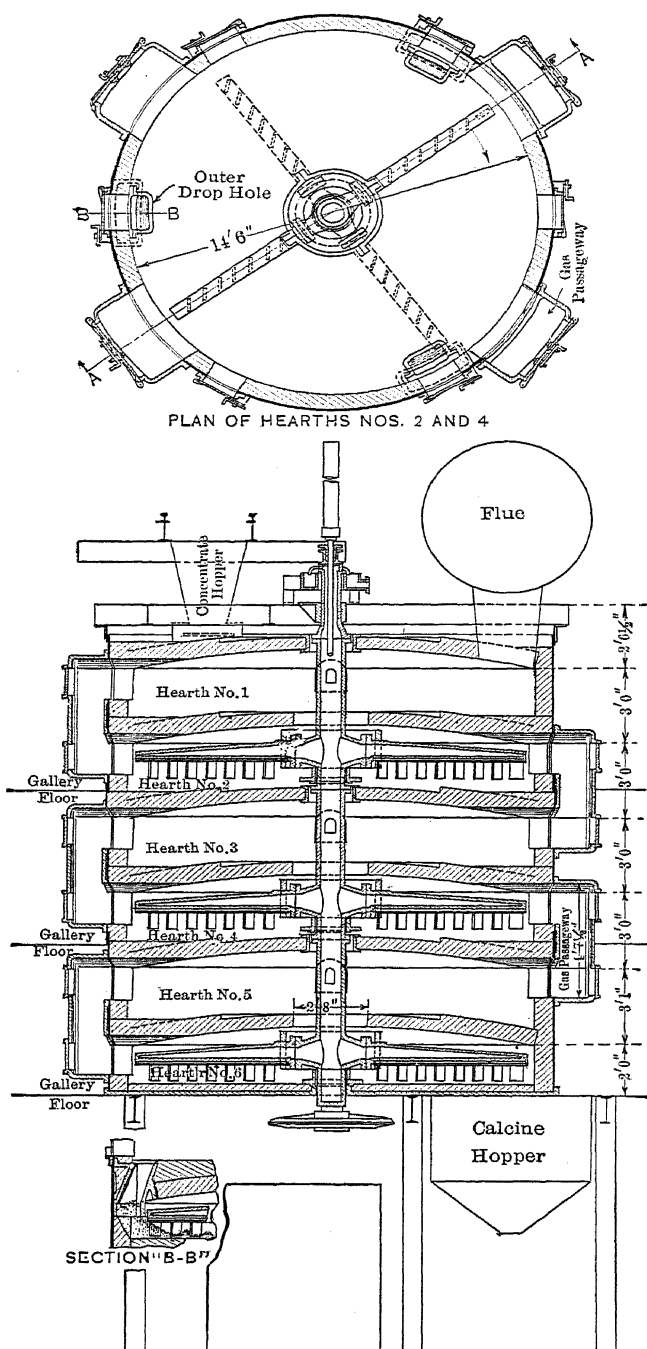
FIG. 4.—REPATH AND MARCY METHOD FOR DUST PREVENTION IN MACDOUGALL FURNACE.

found impossible to operate the furnace without the use of coal and oil. Different conditions of tonnage and draft were tried, but the result was always the same : The furnace would not run without fuel. Even when the fourth hearth was kept hot with coal and oil the calcine produced would assay from 18 to 20 per cent. in sulphur. The test showed, therefore, that the furnace as first constructed was not a success.

From the above it will be seen that the furnace was unsatisfactory on account of its reduced power to eliminate sulphur. Not enough sulphur was burned to either keep the furnace hot or produce desirable calcine. The reason for the low elimination of sulphur was that one of the principal means of oxidizing the sulphur in the Evans-Klepetko type of MacDougall furnace—namely, the showering down of material through the rising current of hot oxygen-bearing gases—had been removed without anything else being substituted to take its place. In the regular type of MacDougall furnace a large amount of sulphur is burned and a considerable proportion of the total heat is generated during the fall of material from hearth to hearth. During the fall each particle is exposed on all sides to the action of oxygen and oxidation goes on much more rapidly than when the particle is lying on the surface of the hearth. Eliminating this fall therefore reduced the amount of flue dust made by the furnace at the expense of oxidizing power.

To make sure, however, that the reduced amount of oxidation was not caused by an insufficient amount of air being drawn through the furnace, a test was carried out under the direction of Mr. Crowfoot to determine the relative amount of gas being discharged by the Repath-Marcy furnace in comparison with two of the regular MacDougall furnaces, Nos. 18 and 1. The test showed that there was about 87 per cent. as much gas by weight being discharged from the Repath-Marcy furnace as from furnace No. 18, and about 81 per cent. as much as from furnace No. 1. The lesser amount of gas discharged by the Repath-Marcy furnace was probably caused by a greater frictional resistance to the passage of air through the furnace in comparison with the regular MacDougall type. The gas passageways, after being lined with brick, had a smaller area than the drop holes in the ordinary MacDougall furnace. Also, the path of the gases through the Repath-Marcy furnace was longer. Inasmuch, however, as the Repath-Marcy furnace was treating less than two-thirds as much material as the other furnaces, it was evident that sufficient oxygen was being drawn through the furnace for the tonnage treated, and that the decrease in power to oxidize sulphur was caused by eliminating the fall of material from hearth to hearth through the rising current of hot gases.

(B). Modified Repath-Marcy Furnace.

Observations on the MacDougall furnaces had tended to show that most of the dust carried into the flue was produced in the fall through the center drop hole from the first hearth to the second, and that a few of the sparks formed in the fall through the side drop holes from the second hearth to the third were also carried into the flue. The test work having indicated that, as then constructed, a certain amount of showering down of material through the rising gases was necessary in order to obtain a good elimination of sulphur, it was decided to modify the design of the Repath-Marcy furnace. The furnace was accordingly shut down and the following changes made : Between the fourth and fifth, and the fifth and sixth hearths the sealed drop holes and separate gas passageways were eliminated, and the construction was changed back to that of the regular MacDougall furnace, allowing the material traveling from the fourth to the fifth and the fifth to the sixth hearths to fall through the rising current of gases as before. Also, pipes were installed to blow compressed air on the fourth and fifth hearths, the equipment being exactly the same as was used in the test to treat a 100 per cent. ore screenings feed.

After the above changes had been made there was no trouble experienced in running the furnace on a feed not exceeding 20 tons per shift, although to maintain the furnace at a working temperature it was necessary to use compressed air continuously. Whenever the air was shut off for a while a cooling action would soon be observed on the third hearth, and in from 2.5 to 3 hr. the calcine would run high in sulphur. Also, it was found impossible to increase the feed over 20 tons per shift without cooling the furnace and producing high sulphurs. The furnace was operated in this way for about a month (from May 16 to June 15, 1911), treating on an average about 18 tons of concentrate, wet weight, per shift and producing calcine averaging about 7.8 per cent. of sulphur.

During the regular operation of this furnace it was desired to compare it with the Evans-Klepetko type of MacDougall furnace, to determine whether or not any improvement had been made in reducing the amount of flue dust produced. As all of the furnaces in each battery discharge their gases into a common flue, any method involving actually collecting and weighing the flue dust would have been out of the question. A method was therefore devised as follows : The two furnaces to be compared were operated on a feed of concentrate only. During the test period all the concentrate charged to each furnace was weighed and sampled, and also all the calcine, calcine barrings, and calcine clean-up produced by each furnace. At the start and at the

end of the test the concentrate hoppers of both furnaces were level full and the calcine hoppers empty. From the weights and assay results, the pounds of copper charged to each furnace and the pounds of copper recovered in the furnace products by each furnace were calculated. For each furnace, the difference between the weight of copper charged and the weight of copper recovered in the furnace products was assumed to be the weight of copper which had gone into flue dust. Applying an average assay for copper in MacDougall flue dust to the above weights of copper assumed as going into the flue, the weight of flue dust produced by each furnace was calculated as shown below:

$$\frac{\text{Pounds copper fed} - \text{pounds copper recovered in furnace products}}{\text{Average assay per cent. copper in MacDougall flue dust.}} + 100 = \frac{\text{Pounds flue}}{\text{dust produced.}}$$

As a check on the above, the weight of flue dust produced was also calculated as follows:

$$\frac{\text{Pounds SiO}_2 \text{ fed} - \text{pounds SiO}_2 \text{ recovered in furnace products}}{\text{Assay per cent. SiO}_2 \text{ in MacDougall flue dust.}} + 100 = \frac{\text{Pounds flue dust}}{\text{produced.}}$$

Using the above-described method, a comparative test was carried out on the modified Repath-Marcy furnace No. 17 and on furnace No. 18 during the 48-hr. period from 9 a.m., May 25, to 9 a.m., May 27, 1911. A summary of the principal data obtained from this test is presented herewith:

Summary of Test Data.

Average.	Furnace No. 18. Evans-Klepetko type.	Furnace No. 17 Modified Repath-Marcy Type.
Tons of concentrate treated per 24 hr. (wet weight), ...	84.96	49.61
Per cent. of moisture in concentrate treated, ...	9.5	9.2
Tons of concentrate treated per 24 hr. (dry weight), ...	76.85	45.02
Tons of calcine produced per 24 hr., ...	46.83	31.67
Assay per cent. sulphur in calcine, ...	10.7	10.7
Per cent. of dry weight of feed recovered in calcine, ...	60.93	70.35
Per cent. of copper fed recovered in calcine, ...	74.3	85.5
Per cent. of sulphur fed eliminated (from calcine), ...	81.5	79.0
Calculated by Cu and SiO₂ methods.		
Pounds of flue dust produced per 24 hr., ...	27,000	9,042
Pounds of flue dust per dry ton fed, ...	351	201
Flue dust, per cent. of dry weight of charge, ...	17.6	10.0

Although the test figures showed that the modified Repath-Marcy furnace as constructed at the Great Falls Smelter made considerably less flue dust, from the fact that the furnace as arranged by us had a very much reduced capacity and could not be operated without the continuous use of compressed air, it was not thought advisable to adopt this type of furnace for regular work.

It should be remarked in passing that the figures for flue dust given above include all of the flue dust actually passing out of the furnaces. Part of this dust settles in the MacDougall cross flues, and from there passes through hoppers and chutes to the calcine hoppers below. Part settles in the main MacDougall flue and is drawn off into cars by the calcine trammers. Practically all of the remainder is recovered in the uptake and crosstake flues and the main dust chamber. Although the MacDougall furnaces produce a large amount of flue dust, therefore, a considerable proportion of the dust is recovered at about the same cost as is involved in handling calcine.

(C). *Crouse Furnace Designed for Dust Prevention.*

Shortly after the completion of the test on the modified Repath-Marcy furnace, a new test was started on a furnace equipped for dust prevention designed by George S. Crouse. The furnace designed by Mr. Crouse was similar to the modified Repath-Marcy furnace in that it was arranged to carry the material from the first hearth to the second and from the second hearth to the third without dropping it through the rising gases, but the method of doing this was entirely different. In the Repath-Marcy furnace drop holes were provided for the descending material and separate gas passageways for the rising gases. In the Crouse furnace the gases traveled upwards through the drop holes exactly the same as in the Evans-Klepetko type of MacDougall furnace. Concentrate passing from the first to the second hearth was discharged through an extra drop hole, which, by an ingenious arrangement, was sealed against the rising gases only during the brief period when material was falling through it. From the second to the third hearth the concentrate fell through the regular side drop holes, these drop holes being open for the rising gases except when concentrate was being discharged through them.

A brief description of the equipment of the Crouse furnace is as follows: The extra first hearth drop hole measured 7 by 8 in. in plan and was located about 4 in. from the edge of the regular center drop hole. All the rakes on the first hearth rabble arms pushed the concentrate towards the center of the furnace except the two inner rakes, one on each arm. These rakes pushed the material away from the center. The concentrate, therefore, instead of showering continuously over the edge of the first hearth center drop hole, as in the regular type of MacDougall furnace, was forced to pass through the exrta 7 by 8 in. drop hole, falling twice in every complete revolution

of the central shaft.   The rabble arms on the first, second, and third hearths were built so as to be always in line one above the other.   On the second hearth rabble arms, towards the center of the furnace, were attached two iron vessels open at the top and bottom, extending from the bed of the second hearth up to nearly flush with the roof. When one of the first hearth rabble arms would be dropping concentrate through the extra 7 by 8 in. drop hole, the receiving vessel on one of the second hearth rabble arms would be directly below the drop hole, sealing it against the rising gases.   Similarly, a receiving vessel was attached near the outer end of each of the third hearth rabble arms, sealing the regular second hearth side drop holes while material was falling through them.

In dropping the material from the third to the fourth hearth, two extra drop holes were used.   This drop hole construction for the third hearth was referred to earlier in this report in connection with the question of enlarging the third hearth center drop hole from 14 in. to 20 in.   The extra third hearth drop holes were not peculiar to the Crouse furnace, being in use in several of the other furnaces at that time.   They were not at any time sealed against the rising gases, but by dropping a large bulk of material four times in every revolution of the central shaft, instead of having it showering continuously over the edges of the center drop hole, the amount of incrustation formed on the third hearth roof was greatly reduced.   Also, the incrustations formed were easier to remove, because they were localized over the two extra drop holes.   These extra drop holes permitted the use of a 14-in. third hearth center drop hole.

One further feature in the equipment of the Crouse furnace was the use of "spark catchers" on the ends of the fourth hearth rabble arms.   These "spark catchers" were plates extending horizontally in front of and in back of the ends of the arms.   The object of the plates was to catch the sparks formed while the rabble arms were pushing material through the fourth hearth side drop holes, thus preventing the building up of incrustations on the roofs.   The plates could be cleaned through one door, and could be cleaned more easily than the roof above the drop holes.

The drop hole arrangement and general equipment of the Crouse furnace is shown in the two accompanying sketches, *A* and *B* of Fig. 5.

The Crouse furnace, then, eliminated the dropping of concentrate through the rising gases between the first and second and the second and third hearths without providing any separate gas passageways. One result of this was that the Crouse furnace was much more compact than the modified Repath-Marcy furnace.   At this point it may
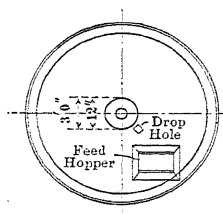
Hearth No. 1.

The center drop hole is now used as a gas passageway only, and extra drop hole for concentrates.
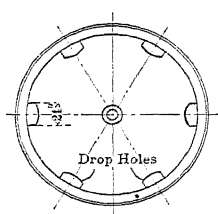
Hearths Nos. 2 and 4.

Usually one or more of these side drop holes are covered to damp the air current.
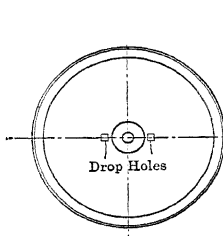
Hearth No. 3.

The two extra drop holes through which the calcines fall to the hearth below, localize and reduce incrustations.
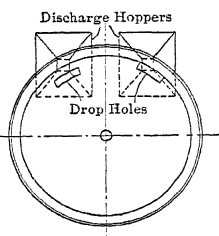
Hearth No. 6.

The discharge drop holes are 8 in. wide and 28 in. long on outer arc.
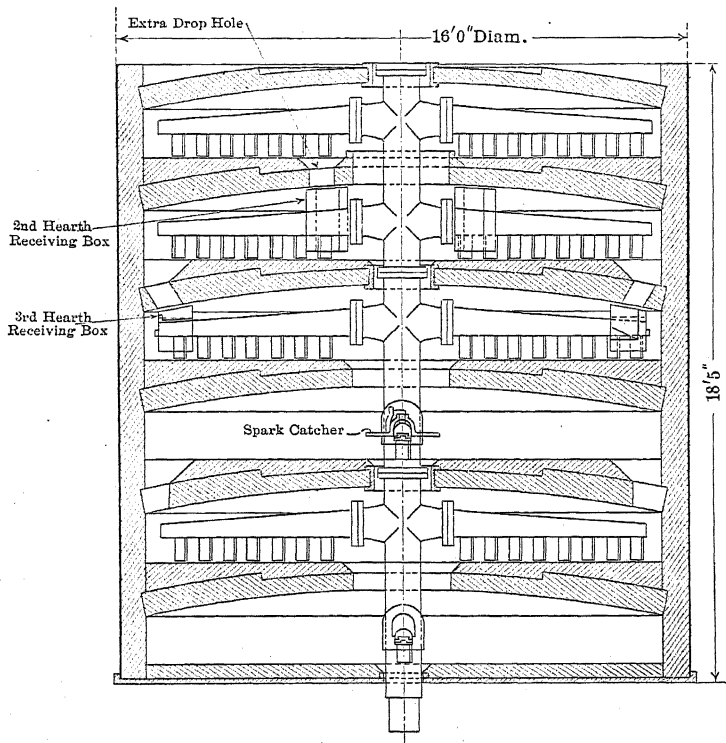
FIG. 5.—CROUSE EQUIPMENT FOR DUST PREVENTION IN MACDOUGALL FURNACE.

be remarked that the dropping of material through the rising gas current may be eliminated between the first and second and the second and third hearths with much less injury to the capacity of the furnace than would be produced by eliminating this drop between the lower hearths. For example, in falling through the rising gases from the first to the second hearths, the concentrate may be dried and heated to a certain extent, but no sulphur is eliminated and no heat generated. In dropping through the gas current between the lower floors, a certain amount of sulphur is eliminated from the calcine and heat is evolved. Dispensing with this drop between the lower floors would therefore not only raise the percentage of sulphur in the calcine, but would also take away part of the means on which the furnace depends for the supply of heat to keep it running. In a furnace where the heat is all produced from the oxidation of the sulphur without the help of any outside fuel, any factor tending to cut down the total amount of heat produced is a serious detriment to the furnace.

(a). *Tests on the Crouse Furnace.*—MacDougall furnace No. 3 was remodeled to conform with Mr. Crouse's design, and was first started on Dec. 18, 1911. From the very beginning this furnace demonstrated that it would run satisfactorily without the use of coal, oil, or compressed air and would produce calcine assaying the same percentage in sulphur as the average of the other furnaces, although at first it was necessary to feed it a slightly lower tonnage than the average of the other furnaces. A number of minor changes were made in the equipment from time to time, such as rebuilding the second hearth drop holes to make them come down more nearly to the tops of the receiving vessels on the third hearth rabble arms, and adding two extensions to the third hearth receiving vessels. In general, these changes were designed to seal the drop holes more securely against the ascending gases during the period of the falling of the material from hearth to hearth.

The first test to compare the Crouse furnace with one of the regular MacDougall furnaces as to the relative amounts of flue dust produced, was carried out in April, 1912. The results of this test showed that the Crouse furnace was producing considerably less flue dust, but at the expense of a certain reduction in calcining capacity. With both furnaces being fed practically the same tonnage of concentrate, 64.5 tons, wet weight, per 24 hr., the Crouse furnace produced calcine assaying 14.2 per cent. of sulphur, as against 10.5 per cent. of sulphur in the calcine from the regular MacDougall furnace. Temperature readings taken to compare the Crouse furnace with the regular MacDougall furnaces gave the following results:

Temperature in °F. on Hearths of MacDougalls.

	Hearth No. 1.	Hearth No. 2.	Hearth No. 3.	Hearth No. 4.	Hearth No. 5.	Hearth No. 6.
Average of six regular MacDougall furnaces, .	429	1,017	1,142	1,216	1,162	1,106
Crouse furnace, ...	560	880	960	1,100	1,135	1,150

From the above figures it will be seen that the Crouse furnace ran considerably cooler than the regular furnaces, as would be inferred from the fact that in the comparative test, treating the same feed as the regular furnace with which it was compared, the Crouse furnace produced calcines higher in sulphur; that is, it burned a lesser amount of sulphur per minute and per square foot of hearth area.[1]

As a result of this test the Crouse furnace was shut down on Apr. 13 and the following changes were made in the equipment:

Changes Made in Equipment of MacDougall Furnace No. 3, Apr. 19 to Apr. 24, 1912.

Hearth.   *Changes.*

First.—Center drop hole increased in size. Formerly measured 11 in. from shaft to edge of drop hole. Changed to 14 in. from shaft to edge of drop hole.

Second.—Side drop holes previously measured approximately 5 by 9 by 20 in. Enlarged by chipping out brick between edge of drop holes and shell of furnace to approximately 12 by 15 by 21.5 in.

Third.—Previously center drop hole measured approximately 20 in. from center shaft to edge of drop hole. Was originally made 11 in. with two extra drop holes, but had become enlarged to 20 in. by the falling out of two center courses of brick, converting the center drop hole and extra calcine drop holes into one large drop hole. Rebuilt, making edge of center drop hole 14 in. from shaft, and two extra calcine drop holes 6 by 7 in.

Fourth.—Side drop holes enlarged from approximately 5 by 9 by 20 in. to approximately 10 by 12.5 by 25 in. Brick chipped out between edge of drop hole and shell of furnace.

Fifth.—No changes. Center drop hole measured 14 in. from shaft to edge of drop hole.

Sixth.—No changes.

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[1] For further data regarding temperatures on the hearths of MacDougall furnaces, see *Practice of Copper Smelting*, by E. D. Peters, pp. 93 to 95, and also paper by L. S. Austin, The Washoe Plant of the Anaconda Copper Mining Co. in 1905, *Trans.*, xxxvii., 431 (1906).

The above changes in general increased the drop hole area, making it possible for a larger volume of air to be drawn through the furnace. Also, the increase in drop hole area was made as far as possible by chipping out the brick between the outer edges of the side drop holes and the shell of the furnace, thus increasing the drop hole area with as little reduction in hearth area as possible. In the regular MacDougall furnaces the standard size of drop hole for the second and fourth hearths is 5 by 9 by 24 in. In comparison with the regular MacDougall furnaces, therefore, the Crouse furnace, after the above changes had been made, had a larger second and fourth hearth drop hole area.

When the Crouse furnace was again started it was found that the changes in construction had brought about a considerable increase in calcining capacity, and that it would handle satisfactorily as large a tonnage as the average of the other furnaces. A comparative test was made between the Crouse furnace and McDougall furnace No. 1, lasting for eight days during the period from May 21 to May 29, 1912. Data obtained from this test are tabulated below:

Summary of Test Data.

Test period: 12 m., May 24, to 9 a.m., May 29, 1912.

Average.	Furnace No. 1.	Crouse Furnace. (No. 3.)
Tons concentrate treated per 24 hr. (wet weight), ...	67.157	68.493
Per cent. of moisture in concentrate treated, ...	8.0	7.9
Tons of concentrate treated per 24 hr. (dry weight), . . .	61.78	63.079
Tons of calcine produced per 24 hr., ...	39.372	42.706
Assay per cent. sulphur in calcine, ...	10.3	9.6
Tons of total products, excluding flue dust, per 24 hr., . . .	40.531	44.334
Per cent. of dry weight of feed recovered in above products, .	65.6	70.3
Per cent. of copper fed recovered in above products, ...	78.6	87.0
Per cent. of sulphur fed eliminated from above products, .	81.3	81.2
Calculated by Cu Method.		
Pounds of flue dust per 24 hr., ...	21,049	13,422
Pounds of flue dust per dry ton fed, ...	341	213
Flue dust, per cent. of dry weight of charge, ...	17.0	10.7

From the above figures it will be seen that the Crouse furnace as finally constructed was a complete success. It had a slightly greater calcining capacity than the regular furnace with which it was compared, and was a big improvement over the regular furnace in that it converted a larger percentage of the feed into furnace products and made less flue dust. Enlarging the drop hole area on the regular furnace to correspond with the drop hole area of the Crouse furnace might or

might not have again given the regular furnace a slightly greater calcining capacity, but the Crouse furnace would still have had the big advantage in its favor of making less flue dust. It may also be remarked here that although on account of the rebuilding of the smelter, and possible changes in the MacDougall plant, the old MacDougall furnaces have not been remodeled to conform with the Crouse design, the original Crouse furnace, No. 3, has continued in successful operation up to the present date.

MacDougall furnace No. 18, with which the modified Repath-Marcy furnace was compared, in the test figures shows a greater calcining capacity than MacDougall furnace No. 1, with which the Crouse furnace was compared. These two MacDougall furnaces, Nos. 18 and 1, are exactly similar in construction, and with the same character of feed have practically equal calcining capacities. A change in the grade of concentrate between the dates of the two tests accounts for the larger tonnage treated by MacDougall furnace No. 18.

X. Concrete Hearths.

A comparatively recent change made in the construction of the MacDougall furnaces, which has tended to decrease the operating cost of the department, has been the substitution of reinforced concrete hearths in place of the old brick hearths. The brick hearths were 9 in. thick, 14 ft. 6 in. in diameter, and were made of regular size bricks laid in regular courses. Due to the expansion and contraction caused by changes in temperature when the furnaces were started and shut down, these bricks would often become loosened and would fall out around the drop holes. Also, bricks would become loosened by barring heavy incrustations from the roofs, and holes would occasionally be formed in the hearths. The falling out of bricks from either of the two above-mentioned causes would decrease the area of the hearths and would allow material to fall through to the hearths below before it had been sufficiently calcined. On this account furnaces would sometimes have to be shut down for repairs after running for only a few days. The hearths would generally last a great deal longer than this, but it was never possible to tell when it would be necessary to close down for repairs.

To avoid the above disadvantages of brick hearths, the substitution of concrete hearths was suggested by Mr. Corwin. As an experiment, the third and fourth brick hearths were removed from MacDougall furnace No. 6 and concrete hearths substituted in their place. The third hearth was selected because it undergoes probably the greatest variation in temperature during regular operation, and the fourth

hearth because it is probably subjected to the greatest variation in temperature on starting up and shutting down the furnace. The material from which the concrete for the hearths was made was as follows :
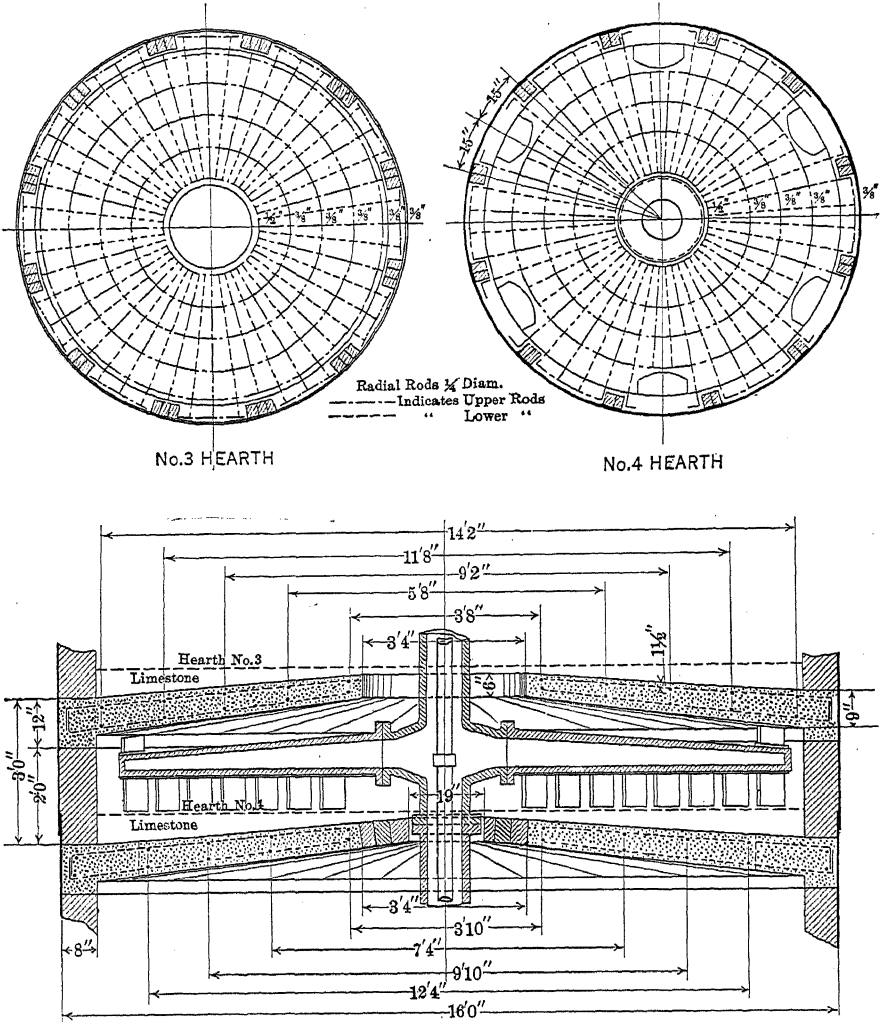
FIG. 6.—CONCRETE CONSTRUCTION OF HEARTH FOR MACDOUGALL FURNACE.

Material.	Amount.
Portland cement, ...	1 part.
Tailings sand, ...	2 parts.
Crushed slag, ...	4 parts.

The hearths were installed by first building wooden forms in the furnace and then putting the concrete in on the forms. These hearths were 9 in. thick where they joined the shell of the furnace, decreasing to 6 in. in thickness near the center shaft. The concrete was reinforced by two series of concentric iron rings tied together by radiating iron rods. One series of concentric rings and radiating rods was buried in the concrete about 1.5 in. above the bottom of the hearths, the other about 1.5 in. below the top. The inner one of the concentric iron rings in each series was made of 0.5-in. iron rod, the two ends fastened together, and the outer rings of ⅜-in. iron rod. The radiating iron rods were all 0.25 in. in thickness, 24 in each series. The method of reinforcing the concrete is shown in detail in Fig. 6.

After putting in the third and fourth hearths of concrete, MacDougall furnace No. 6 was started on July 24, 1912. In November the furnace was shut down and the second and fifth hearths changed from brick to concrete, the furnace being started again on Nov. 23, 1912. From November, 1912, to the present date (May, 1913), the furnace has been in continuous operation except for short shutdowns to repair the machinery, etc. The hearths are still in as good condition as when first installed, showing no cracks or wear of any kind. The heavy incrustations that form on the roofs of the hearths are removed more easily, as they do not adhere as tightly to the smooth surface of the concrete as to the rough surface of the brick hearths. The hearths give every appearance of being practically indestructible, and the indications are that the MacDougall furnaces, when constructed throughout with concrete hearths, will run practically independent of masonry repairs.

XI. CONCLUSION.

As has been stated, systematic experimental work to increase the efficiency of the MacDougall furnaces was first started in May, 1910. Herewith are given in parallel comparison data on tonnage treated in the MacDougall department for the months of April, 1910, and April, 1913. Some figures are given showing the reduction in labor required for operating the department. The figures on labor do not include foremen, trammers, oilers, samplers, and so on, as the number of men employed in such positions is practically independent of the capacity of the furnaces. The figures include only the occupations directly connected with the running of the furnaces where a decrease in the number of furnaces in operation means a decrease in the number of men required.

	April, 1910.	April, 1913.
Number of days McDougall department in operation,	30	30
Total tons cupreous material treated during month, .	17,778.4	18,369.4
Number of furnace days, ...	386.9	238.4
Average number of furnaces in operation per day, .	12.9	7.95
Tons cupreous material treated per furnace day, . .	46.0	77.1
Average per cent. S in calcines, ...	7.9	8.9
Partial List of Labor used in Operating the McDougall Department		
Total number of furnacemen during month, ...	270	180
Average number of furnacemen per 24 hr , ...	9	6
Average number of furnacemen per 100 tons cupreous material treated, ...	1.52	0.93
Total number of feeders during month, ...	105	80¼
Average number of feeders per 24 hr., ...	3½	2⅔
Average number of feeders per 100 tons cupreous material treated, ...	0.59	0.44
Total number of laborers pulling lumps during month,	90	. . . .
Average number of laborers pulling lumps per 24 hr.,	3	. . . .
Average number of laborers pulling lumps per 100 tons cupreous material treated, ...	0.51	. . . .

From the above figures it will be seen that the MacDougall department treated a larger tonnage of cupreous material in April, 1913, with an average of 7.9 furnaces operating per day, than in April, 1910, with an average of 12.9 furnaces operating per day. Due to the fewer furnaces running in April, 1913, there was a saving in labor over April, 1910, of three furnacemen per 24 hr., three laborers pulling lumps per 24 hr., and ⅚ feeder per 24 hr.

The experimental work in connection with the MacDougall department resulted not only in increasing the capacity of the furnaces, but also in increasing the percentage in first-class ore screenings which could be handled in the feed. It brought about the development of a furnace producing a much smaller percentage of flue dust, and the substitution of reinforced concrete in place of brick hearths. All of the above changes tended to decrease the daily operating expense of the department. These improvements were made without building any new furnaces or enlarging the old ones, and are interesting as showing what can be accomplished by systematically studying the conditions governing a smelting department.

Valuable general information about MacDougalls and about MacDougall practice at the Great Falls and Washoe smelters will be found in the *Practice of Copper Smelting*, by E. D. Peters, and in the following papers :

The Washoe Plant of the Anaconda Copper Mining Co. in 1905, by L. S. Austin, *Trans.*, xxxvii., 431 (1906).

Progress in the Metallurgy of Copper, by L. S. Austin, *Mineral Industry*, vol. xv. (1906).

Notes on the Metallurgy of Copper in Montana, by H. O. Hofman, *Trans.*, xxxiv., 258 (1903).

The Development of Blast-Furnace Construction at the Boston & Montana Smelter.

BY J. A. CHURCH, JR., GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

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I. EARLY FURNACES.

COPPER blast-furnace construction in America has long recognized a general standard in the rectangular water-jacketed shaft with separate forehearth. The details, however, and especially those relating to shaft dimensions, have had to meet a great variety of conditions, and show no uniformity whatever. Selecting a few of the present day Western plants, we find furnace lengths varying from 12.5 to 87 ft., widths at tuyere level from 42 to 84 in., and heights above tuyeres from 5 ft. 4 in. to 22 ft. 4 in.; and comparing present designs with those of the past, we meet contrasts no less striking. Unlike its transverse section, the length of a furnace has come to depend rather on the size of unit desired than on the requirements of smelting; and if the question of length be ignored, several of the various existing designs can be grouped together on the basis of a common transverse section. This section, having a width at tuyere level of 56 in. and a height above tuyeres of 18 ft., was developed at Great Falls, and the story of its development contains some interesting features.

Ground was broken for the Great Falls works in the spring of 1891. About a year later, the concentrator was able to begin operation, followed within the next few months by the Brueckner, reverberatory, and converter departments, and in April, 1893, by the first blast fur-

naces. These, with additions completed in 1894, included four units, having a total daily capacity of about 500 tons. Nos. 1 and 2 were old furnaces, having already seen service at Butte; their hearth dimensions were 42 by 84 in. Nos. 3 and 4 were new. No. 3 was 120 in. long, 36 in. wide at the bottom of the jackets, $39\frac{5}{8}$ in. wide at the tuyeres, and 8 ft. 1 in. high from tuyeres to charge floor (see Fig. 1). The crucible formed part of the water-jacketed shaft, the main jackets extending 27 in. below the tuyere level and having a recess cut at one end for the breast jacket. This general construction, which was abandoned in the next furnace built, has reappeared after 17 years in the present No. 3, though the breast jacket, as in all the latter furnaces, is at the side. There were seven tuyeres on each
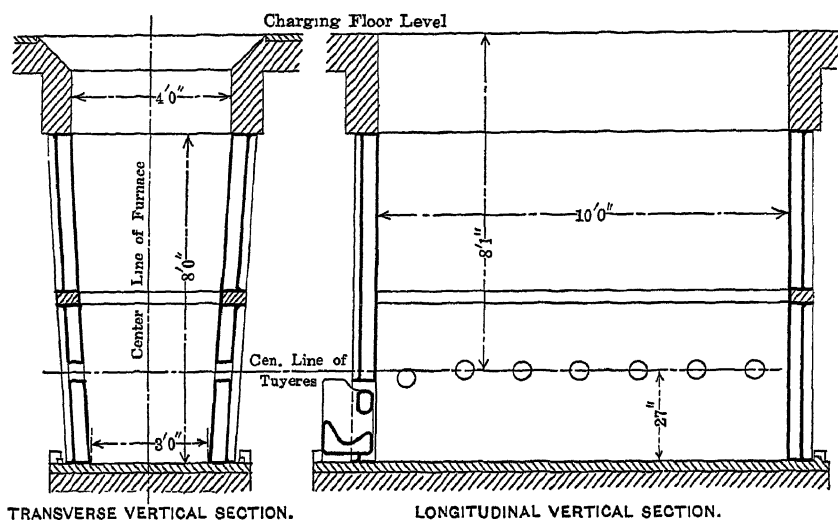
FIG. 1,—36 BY 180 IN. BLAST FURNACE. ERECTED AS NO. 3 OF THE ORIGINAL BLAST-FURNACE PLANT.

side, that nearest the discharge end being placed 2 in. lower than the others. No. 4 was 90 in. long, 34 in. wide at the bottom of the jackets, $36\frac{1}{2}$ in. wide at the tuyeres, and 8 ft. 2 in. high from tuyeres to charge floor (see Fig. 2). The jackets, extending 11 in. below the tuyere level, rested on the walls of a brick crucible 12 in. deep, the breast jacket being set in the brick under one of the end jackets. This became the standard type of crucible, from which no departure was made until the year 1911. The tuyeres were 12 in number, 6 on each side.

The foreharths of the furnaces were of the movable type common at the time. That designed for No. 3 furnace consisted of a cast-iron body lined with brickbats, and mounted on a wheeled truck, its out-

side dimensions being roughly 5 ft. length, 3 ft. width and 2 ft. depth. When a lining gave out, the whole affair was rolled out of the way and replaced by another, freshly lined.

At this time the blast furnace was still something of an experiment with Butte ores, and the early practice was with small furnaces and low blast pressures. A majority of metallurgists in the Butte district preferred the reverberatory, and, with blast-furnace charges carrying as much as 90 per cent. of fines, this preference was not remarkable. In spite of low volumes and pressures of blast, the production of flue dust was excessive; hand feeding was the rule, and the use of lime was looked upon with some suspicion. Nevertheless, the process of raw-ore smelting had already made a beginning, and it needed only a gradual decline in the grade of ore to bring practice
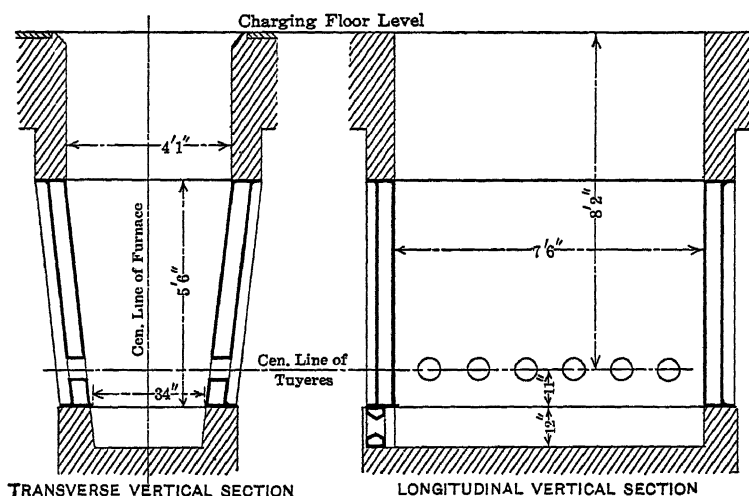
FIG. 2.—34 BY 90 IN. BLAST FURNACE. ERECTED AS NO. 4 OF THE ORIGINAL BLAST-FURNACE PLANT.

up to its present level. These were the conditions under which the Great Falls furnacemen began their work, and out of which they produced a design of furnace that has contributed very largely to the blast-furnace production, not only of the district, but of the country.

The first step in this development was the substitution of stationary settlers for the movable forehearths. This was tried in the spring of 1894, and proved an instant success. The first settler was built for Furnace No. 1, with a diameter of about 7 ft., but within a short time all the furnaces were similarly equipped.

A record of the practice at this time will be found in Table I., which gives the tonnage and composition of charge for each furnace

during August, 1894. It will be noted that Furnace No. 1 was running chiefly on re-treated material, 97 per cent. of its charge consisting of matte and converter slag. No assays appear for this month, but an assay of first-class ore, dated in October of the same year, shows a copper content of 25 per cent.; and another of coarse concentrates, of the same date, shows 20 per cent. of copper.

In April, 1895, a test was run on the Charles Allen process of Bessemerizing in the blast-furnace crucible. ·The crucible of furnace No. 3 was equipped with a second set of tuyeres, connected with the converter blast main, to raise the grade of the normal furnace matte as it came down from the smelting zone above. The experiment was a failure, for it gave uncertain results and did a good deal of damage to the jackets, which under these new conditions quickly burned through; but it furnished some interesting figures, which will be found in Table II. The first two runs show the desired increase in the grade of matte; and the second, in which slag and matte were sampled together, shows a corresponding fall in the silicate degree and a rise in the copper content of the slag. These changes evidently mark an approach to converter conditions. The third run, on the other hand, gave no results whatever; blowing for 1.5 hr. raised the grade of matte only 0.3 per cent. Whatever the reason for this may be, it is not shown by the records; but the fact itself helps to explain why the work was dropped.

The regular practice in 1895 and 1896 is shown in Table III. It will be noted that furnace No. 1 was still re-treating large quantities of matte and slag, and that its percentage of fuel is correspondingly low. Of the remaining three furnaces, No. 3 as the largest furnace smelted the largest gross tonnage, and No. 4 showed the greatest tonnage per square foot of hearth area. The fuel burdens of these three furnaces were almost exactly the same, the weighted averages for all months in which comparisons could be made being: for No. 2, 10.14 per cent.; for No. 3, 10.17 per cent.; and for No. 4, 10.15 per cent.

II. Experiments with the High-Shaft Furnace.

In the spring of 1897 the original No. 1 furnace was replaced by an entirely new design. The high cost of using barren limestone to flux the siliceous Butte ores had already made itself felt, and an effort was now made to replace more of this lime with ferrous oxide derived from the ore itself. To increase the formation of ferrous oxide, it was proposed to expose the charge to furnace action for longer periods of time, and with this idea a very considerable increase was made in

the height of shaft.   As rebuilt, No. 1 was 24 ft. 5 in. high from tuyere level to charge floor.   The jacketed portion of this height was covered by two tiers of jackets (see Fig. 3), of which the lower extended 11 in. below the tuyeres and rested on the walls of a brick crucible, thus following the crucible construction of old No. 4.   The upper side jackets were vertical, giving a shaft width of 48 in.; the lower tier was given a slight bosh, which reduced this width to 42 in. at the bottom of the jackets and 42.5 at the tuyere level.   The end jackets were vertical, 180 in. apart.

The furnace was blown in on July 16, 1897.   The first difficulty to appear was a lack of trap in the spout; but this was soon remedied, and a number of campaigns were then made with fair results. However, other and more serious troubles made their appearance,
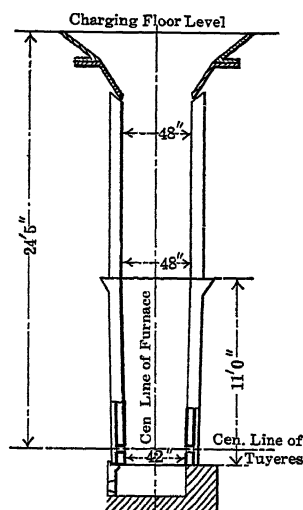
FIG. 3.—END ELEVATION OF NO. 1   LAST FURNACE.   ERECTED IN 1897.

and were so persistent that the type was finally abandoned.   The most interesting of these troubles, and the one most closely connected with the construction of the furnace, was a decrease, instead of the expected increase, in the degree of oxidation obtained.   Any attempt to keep the charge column up to the full height of the shaft always lowered the grade of matte, and the furnace was usually run with a comparatively low column.   Although the blast pressures on No. 1 were much higher than on the others, these difficulties were ascribed to insufficient blast, and the high column was given a final test at the full capacity of the blowing plant.   For this purpose, the other furnaces were temporarily shut down, and by shutting off some of the tuyeres on No. 1 the blast pressure was forced up to a maximum of 60 oz.

Nevertheless, the results were unsatisfactory, for the grade of matte decreased as before.

These difficulties of course indicate a lack of oxygen. When the column was raised its resistance to blast was increased; and while the pressure might rise, the volume of air fell off. This rule held good, not only in normal running, but also in the special test, when the other furnaces were shut down; though in the latter case the volume of blast was also decreased by reducing the tuyere area. If all the tuyeres had been left open enough blast might have been obtained to secure the desired oxidization, although, with the type of blower then in use, an increase in height of column would always tend to raise the pressure and lower the volume received by the furnace. It appears that the high column in No. 1 was never tested under adequate blast conditions, and that its results were inconclusive.

This does not mean that the design of No. 1 as a whole was good, even as an example of the high-shaft furnace. It was too narrow between upper jackets, and the formation of crust, which was, of course, common enough, often led to bridging near the top of the column. Once formed, the crusts and bridges were hard to break, for the low columns and narrow shaft combined to make barring difficult.

There was another difficulty, due more to the experimental nature of the furnace than to its actual design. All the furnaces had a common charging floor; No. 1, therefore, was much lower than the others at the tapping level, and consequently the matte ladle had to be set in a pit several feet below the normal level of the converter floor. This pit collected water, and in case of a runaway was always a source of danger.

When it was decided to remodel the furnace, the question of greatest width was being tested on Nos. 2 and 3 and no conclusive results had been obtained. Moreover, it was desired to remodel as quickly as possible, so the upper part of the shaft retained its width of 48 in., and the width at the tuyeres was merely increased from 42.5 to 44 in. The height was reduced to about 18 ft., to conform with Nos. 2 and 3. No. 1 was remodeled on these lines during the spring of 1899, and in its new form was blown in on June 8 of the same year. This type needs no further discussion, because it was more or less of a makeshift, and stands apart from the regular development of the Great Falls furnaces.

The record of No. 1 as the high-shaft furnace is summarized in Tables IV. and V. For comparison, the records of Nos. 3 and 4 are given in Table VI. for the same period as that covered by Table V.

It will be seen that during the latter part of 1898 the matte from No. 1 was slightly higher in grade than that from Nos. 3 and 4; but it must be remembered that No. 1 was usually run with a comparatively low column, so the results do not properly represent the high shaft.   In fact, Table VI. shows that the height of column during this period was about 15 ft., or 9 ft. less than the height of the shaft.

Following the usual practice at Great Falls, the oxygen ratios given for the slags in Tables IV., V., and VI. were calculated on the assumption that alumina in the blast furnace acts as an acid.   Whatever truth there may be in this assumption, it has been used in calculating all the oxygen ratios given in the tables, and the results are therefore fairly comparable.   If it be true, most of the slags produced in these earlier years were abnormally acid, compared to present practice; and those from No. 1, shown in Tables IV. and V., were no exception.

III. EXPERIMENTS WITH THE WIDE FURNACE.

Experience with No. 1 during 1897-98 pointed towards a lower and wider furnace.   A reduction in height would remove what seemed to be a useless part of the shaft, and would avoid tapping into a pit below the converter floor level.   An increase in width between upper jackets would avoid the sticking and bridging so common in the narrow shaft of No. 1.   Accordingly, during the year 1899 two new designs were made, for furnaces 17 ft. 11$\frac{3}{4}$ in. high from tuyere level to charging floor, 180 in. long between end jackets, and 72 in. wide between upper side jackets.   Just why the foregoing height was chosen does not appear; but it probably was governed chiefly by the available head room, after allowing for the desired change in tapping level.   The decision as to width was a matter of judgment, guided by experience with No. 1.

There remained the question of width at the tuyeres.   Furnaces 36, 39, and 42 in. wide at this point had given good service, and there was no reason to suppose that the limit had been reached.   Accordingly, the whole subject was tested by making the new units of different widths at the tuyere level, both greater than anything tried up to that time.   No. 2 was built on its present lines, with a width at the tuyeres of 56 in. (See Fig. 4.) No. 3 was given a corresponding width of 72 in., the side jackets being vertical from top to bottom. (See Fig. 5.)

It is evident that No. 2 was a compromise between the extremes of Nos. 1 and 3, and that the more typical results for a wide furnace were to be expected from No. 3.   The record of No. 2 is therefore taken up later, to show its survival as a standard type.

No. 3 was blown in with the 72 by 180 in. hearth section on Mar. 24, 1899. The furnace ran only about nine months, and in that time developed operating troubles which finally caused a complete change in design. One of these troubles was the production of sow. Another
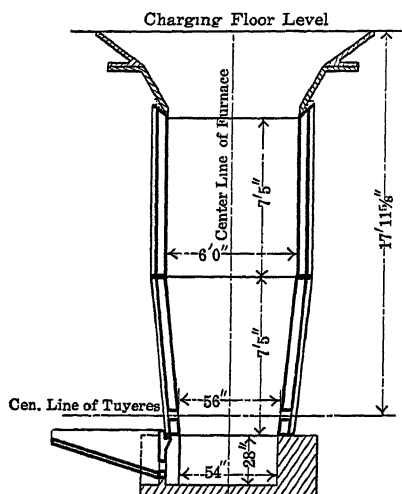
FIG. 4.—TRANSVERSE VERTICAL SECTION OF 56 BY 180 IN. BLAST FURNACE. ERECTED IN 1899, AS No. 2.
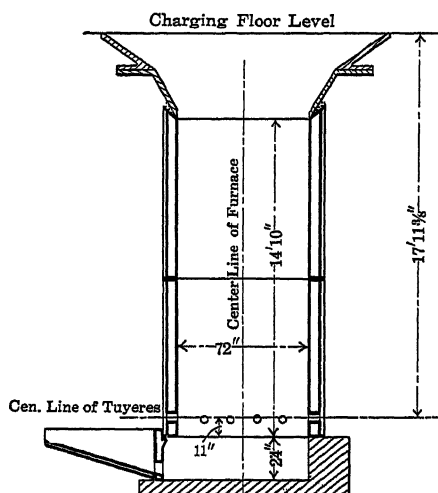
FIG. 5.—TRANSVERSE VERTICAL SECTION OF 72 BY 180 IN. BLAST FURNACE. ERECTED IN 1899, AS No. 3.

was the tendency of the blast to rise along the jackets, instead of spreading evenly through the charge column; and with this condition there appeared heavy, loose banks, or crusts, on the sides, which made the furnace hard to handle. A third difficulty, of less importance

than these other two, was the relatively low temperature of the stream flowing from the spout. The worst of these troubles seems to have been the production of sow; at times it caused the freezing up of the furnace, and at others interfered with the running of the settler, closing the tap holes and making it necessary to tap at a higher level.

In spite of these operating difficulties, No. 3 was very little behind No. 2 in actual results. The detailed records of the two furnaces will be found in Tables VII. and IX., from which the following summary has been taken for purposes of comparison. It should be said that No. 2 also suffered from uneven blast and production of sow, though not so much as No. 3.

Period.	April–June.	July–September.	October–December.
No. 2 Furnace.			
Percentage composition of charge :			
Converter slag,	23.9	23.6	20.3
Concentrates,	24.2	18.2	20.7
Ore,	26.5	34.2	38.1
Limestone,	24.9	23.4	25.0
Tons smelted per operating day,	441	391	350
Tons per operating day per sq. ft. hearth,	6.3	5.6	5.0
Percentage of fuel,	8.3	9.3	9.5
Blast pressure, ounces,	35	33	33
No. 3 Furnace.			
Percentage composition of charge :			
Converter slag,	26.1	24.0	21.7
Concentrates,	26.7	19.5	22.1
Ore,	23.3	33.2	31.1
Limestone,	23.3	23.0	23.8
Tons smelted per operating day,	350	420	312
Tons per operating day per sq. ft. hearth,	3.9	4.7	3.5
Percentage of fuel,	8.9	9.3	10.2
Blast pressure, ounces,	33	33	32

These figures show that the two furnaces ran on practically the same charge. Since both were connected to a common blast main, the blast pressures are naturally almost identical. In gross tonnage per day, No. 2 exceeded No. 3 in the first and third periods, but fell behind in the second; and for the three periods combined No. 2 is again slightly in the lead. In tonnage smelted per square foot of hearth No. 2 always exceeds No. 3. In the first and third periods No. 2 was slightly more economical in fuel; but in the second, when No. 3 showed a better daily tonnage, the two furnaces in this respect were exactly alike.

It appears, therefore, that the wide form was not abandoned on account of its record, which was very nearly as good as that of the form

which superseded it.   No figures are available to show the relative volumes of air received by Nos. 2 and 3, but their similarity in daily tonnage, in blast pressures, and, as far as recorded, in heights of column, suggests that these volumes were not very different.   That is, the records of the two furnaces seem to have been controlled by some factor other than furnace design, and of all external factors, the blast conditions are usually the most important.   There is some reason, therefore, to suppose that the daily tonnages of both No. 2 and No. 3 merely kept pace with the volumes of air supplied, and that on this account No. 3, as the larger furnace, showed a lower tonnage per square foot of hearth.   The production of sow suggests that the action of the furnace was not strongly oxidizing; and with more air No. 3 might have shown improvement, not only in capacity, but also in ease of handling.   The results of this form were therefore inconclusive, in so far as they related to a width of 72 in. at the tuyeres.

On the other hand, nothing seems to have been gained by making the sides vertical throughout.   A cool stream at the spout implies a low temperature at the melting zone; and, though various factors may have helped to produce this effect, the experience of the plant has been that sharper smelting action usually follows a contraction of the lower jackets.

These conclusions agree fairly well with the results of a still wider furnace, the present No. 3, which is described farther on.   This new design, which is 84 in. wide at the tuyeres, differs from the 72 by 180 in. type in almost every feature aside from width, having greater height, a sharp bosh in the lowest jackets, and a blast supply more than adequate to its needs.   The good results it has given show that the 72 by 180 in. furnace had by no means reached the limit of width.

IV.   EXPERIMENTS WITH EXTREME BOSH.

In January, 1900, No. 3 was remodeled, with the idea of combining the good points of Nos. 1 and 2.   No. 1, which was 44 in. wide at the tuyeres and 48 in. wide between upper jackets, was usually in good condition at the smelting zone, but gave a good deal of trouble by crusting and bridging over on top.   No. 2, which was 56 in. wide at the tuyeres and 72 in. wide between upper jackets, was easier to keep open on top, but did not handle its blast as well in the smelting zone.   The new furnace was therefore built 72 in. wide between upper jackets and 44 in. wide at the tuyeres, making the bosh very marked (see Fig. 6).   The height was about 18 ft. and the length 180 in., as before.

On Jan. 18, 1900, the furnace was blown in.   The campaign lasted

only three months, and gave such poor results that it was never re-
peated.   During February and March the ore supply carried a good
deal of fines, and at times all the furnaces suffered; but No. 3 was in
trouble nearly all the time, and had the poorest record of all.   On
Apr. 24 the crucible broke out at the back, and advantage was taken
of this accident to shut down the furnace and rebuild it on the lines
of No. 2.   In its new form, No. 3 was blown in again on Apr. 29,
1900, and thereafter gave good results.

   With the narrow hearth and sharp bosh, No. 3 developed troubles
almost exactly the opposite of those which marked the wide hearth
and vertical sides.   It discharged a hot stream at the spout, and made
almost no sow, but the crusts were so heavy and so hard that the fur-
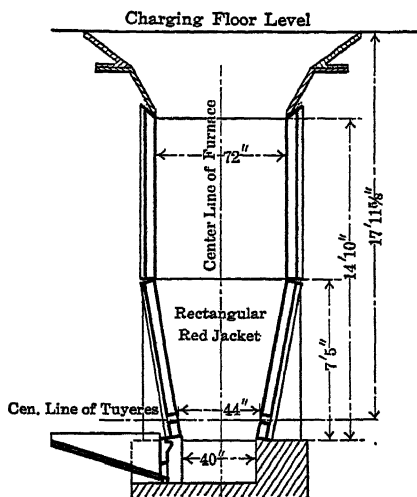
FIG. 6.—TRANSVERSE VERTICAL SECTION OF 44 BY 180 IN. BLAST FURNACE.
ERECTED IN 1900, AS No. 3.

nace could scarcely be kept open on both ends at once.   Some of
these results, high smelting temperature and absence of sow, had been
expected, but the excessive crusting was a surprise, for the furnace
had been designed to gain all the advantages of No. 1 at the hearth,
without its tendency to crust on top.   As far as the hearth conditions
were concerned, the design was a success.   Smelting temperatures
were high, and the absence of sow suggests no lack of oxidation.
However, the extreme hardness of the crusts shows that the high
temperature of the smelting zone was reflected in the upper part of
the column, to an extent never found in either No. 1 or No. 2.   Active
smelting apparently began at a higher level in No. 3; and from a
comparison of the three types, this seems to have been due to the

rapid convergence of the lower jackets, which tended towards an earlier concentration of smelting action. The flat slope of the jackets also gave some mechanical support to the crusts, making them harder to handle. As far as the records go, therefore, the design failed on account of inherent defects.

In the present No. 3, described later, the bosh of the lowest jackets is again very marked, but it extends a much shorter distance above the tuyeres, and represents a much smaller proportion of the total height. It seems merely to intensify smelting action at the proper level, without having much effect on the column above.

Table VIII. gives the record of No. 3 for January, February, and March, 1900. In per cent. of coke and tonnage per square foot of hearth, the furnace almost exactly equaled No. 2 for the same period; its gross tonnage, of course, was less.

V. SURVIVAL OF THE 56 BY 180 IN. FURNACE.

At the end of April, 1900, the blast-furnace department comprised three units. In hearth section, No. 1 was 44 by 180 in., and Nos. 2 and 3 were 56 by 180 in.; all three had a height of 17 ft. 11⅝ in. from tuyeres to charge floor. A fourth furnace was blown in on Oct. 2, 1900, and a fifth on Feb. 22, 1901, both conforming to the 56 by 180 in. type. Finally, in January, 1905, No. 1 was altered to the same lines, and started again on Aug. 24 of the same year.

The type first represented by No. 2 was thus the only one to survive. The causes of its survival may be found in the conditions, especially those of blast, under which the furnaces were operated. Until 1911, testing was confined almost wholly to furnace design, and the blast supply received less attention. The 56 by 180 in. type was therefore the product of a special set of conditions, in which a radical change might have developed a very different style of furnace. Lately steps have been taken towards new methods and new conceptions in the matter of blast supply, and one of the first results has been the successful operation of a furnace which differs in almost every essential from the older type.

This does not mean that the question of blast has been neglected. The early furnaces were run on 6 or 8 oz. pressure, in accordance with the practice of the time. The advent of larger units called for higher pressures, and No. 1, when it had the high shaft, was run on from 22 to 40 oz. (see Tables IV. and V.). During the same period pressures were also higher on the older furnaces, Nos. 3 and 4 being run on from 10 to 18 oz. (see Table VI.). During 1899 and 1900, when three large units were in operation, the pressures varied from

26 to 36 oz. Since then, in spite of the addition of two more units, they have risen to 40 and 44 oz. These figures show that the blast supply has been gradually increased, but that a good deal of the increase has come since the 56 by 180 in. type became established. The real advance in methods of supplying blast was made when the oxygen of the air came to be regarded as part of the furnace charge, and a definite volume of blast was maintained, irrespective of the pressure involved.

The bad points of the other types show why No. 2 met its working conditions so well. It was 6 ft. lower than the high-shaft furnace, and thus offered less resistance to the passage of blast; it received more air, therefore, at the pressure applied. It was 24 in. wider than the same furnace between upper jackets, and so avoided a good deal of trouble in handling crusts. It was 16 in. narrower than the wide furnace at the tuyeres, obtaining a better distribution of blast and better conditions in the smelting zone, and its reduction in width at the tuyeres was not enough to form a very pronounced bosh, so that the troubles of No. 3 in its 44 by 180 in. form were not repeated. In comparison with No. 2, all the other types were extreme in some particular feature, and No. 2 itself was a compromise among all of them.

No. 2 improved upon the other types not only in ease of handling, but also in capacity. During the period from April to December, inclusive, 1899, it treated 393 tons of charge per day, excluding fuel, against 364 tons treated by No. 3 in its 72 by 180 in. form. During January, February and March, 1900, it treated 360 tons, against 273 tons treated by No. 3 in its 44 by 180 in. form. The high-shaft furnace was torn down about the time No. 2 was started, but its tonnage during the last four months of 1898 was only 282 tons per day.

Various minor improvements have been made in the 56 by 180 in. furnace since it was first designed, but only one of these is of much importance. Up to 1911, the floor of the crucible was always built below the breast opening, forming a shallow well at the bottom of the furnace. Extra tap holes were left at the ends and back, entering this well just above the floor, and if tapping out was necessary, the contents of the crucibles were run into earth beds behind the furnace, to be broken up and wheeled away when cold. The furnace spout decreased in depth towards the outer, or discharge, end, so that nothing could be run out through the breast opening unless there was blast enough to overcome the trap (see Fig. 6). When the present No. 3 was designed, the crucible floor was sloped downward from all directions towards the breast opening, which was thus at the lowest point in the furnace. The spout increased in depth outward from the breast,

and its discharge end was covered by a thick plate, the lower portion of which was pierced near the bottom by a tap hole (see Fig. 7). To tap out the crucible, it was necessary simply to open this tap hole and run the molten material through the spout into the settler, where it belongs. This design, which was borrowed from another plant, not
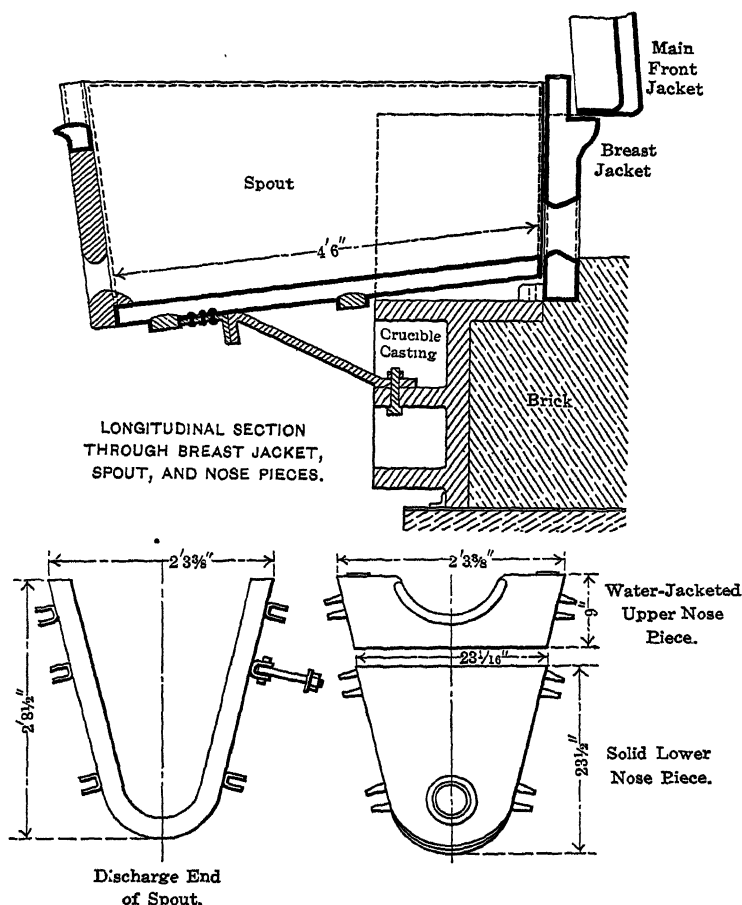
FIG. 7.—BLAST-FURNACE SPOUT INSTALLED ON FURNACE NO. 2 IN 1912.

only does away with bedding the slag, but increases the temperature of the stream running from the spout, for the molten material escapes from the crucible as fast as it formed, and undergoes no period of stagnation and cooling below the breast opening. Since its trial in No. 3, some of the 56 by 180 in. furnaces have been altered to the same design.

VI. RECENT EXPERIMENTS.

Under the old conditions of blast supply, the 56 by 180 in. type became the standard of the plant, and from 1905 to 1911 it was the

only style of furnace in use. In the latter year, however, experiments were made with a new type of blower and a new design of furnace, with the idea of developing a larger transverse section, which should utilize more air and smelt more material, without calling for an increase in labor costs. The full discussion of this new design is reserved for the future, but a brief description can be given here.

Furnace No. 3, which was selected for the experiment, was rebuilt in 1911. Its length is 180 in., as before, the novelty of the design referring entirely to the transverse section. Its height is 22 ft. 4 in. from tuyeres to charge floor, and includes three tiers of jackets. It is 84 in. wide at the tuyeres, 120 in. wide at the top of the lowest tier of jackets, and 96 in. wide at the top of the uppermost tier of jack-
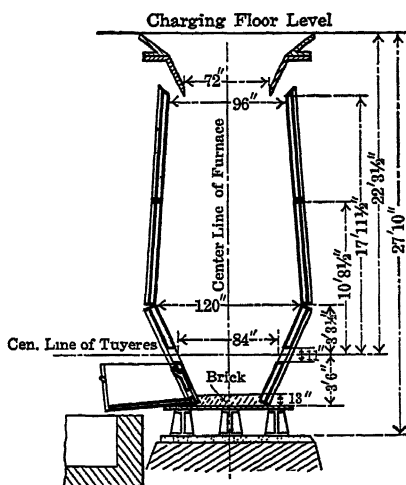
Fig. 8.—Transverse Vertical Section of 84 by 180 In. Blast Furnace. Erected in 1911, as No. 3.

ets (see Fig. 8). The sides are thus boshed in two directions, the lower or normal bosh being confined to the lowest jackets, and the reversed bosh extending over the two upper tiers; the normal bosh is very marked, the reversed bosh more gradual. The lowest jackets rest on cast iron plates, supported by short cast columns. These plates are covered within the furnace by a thin layer of brick, sloped downward towards the opening in the breast jacket, which is set in a recess cut in one of the side jackets. The tuyeres are placed about midway up the lowest tier, the water-jacketed space underneath serving as a crucible; it will be remembered that the essential features of this construction were used in one of the early furnaces, the original No. 3. The tuyeres, 56 in number, are in the form of vertical slots, 11 in. long and 3 in. wide.

The furnace is supplied with blast by a turbo-blower of high capacity, equipped with a constant-volume governor; the flow of air through the tuyeres is thus kept constant in volume, without regard to the pressure involved. There is a limit, of course, to the pressures at which the machine will work, but at the required volumes, from 17,000 to 20,000 cu. ft. of free air per minute, this limit is never approached.

The furnace spreads its blast evenly, shows some economy in coke, and under normal conditions discharges a hot stream at the spout. It has made some record runs, maintaining a capacity for short periods as high as 7 tons per square foot of hearth per day. The charge column, although sometimes raised nearly to the full height of the shaft, is usually held a few feet lower. One object of this practice is to obtain a better distribution of the charge, as it falls from the cars. All the furnaces are fed from the side, but in No. 3 the charging plates overhang the shaft for a distance of 12 in. (see Fig. 8), so that there is a narrow strip along each side which cannot be fed higher than the lower edge of the plate. At the height of column ordinarily adopted, however, this arrangement of the charging plates helps to keep the center level with the sides, and in fact is a step towards overhead charging.

VII. SUMMARY.

The results of these experiments in furnace construction are difficult to summarize, because they have developed no limiting dimensions. However, they point out certain tendencies in operation which must be remembered, if the size of furnace is to be much increased. They also show the importance of the blast, and open up a field for future research in methods of supplying and controlling the air.

In general, an increase in height of column calls for higher blast pressures, if the volume of blast is to remain unchanged. The cost of air compression rises with the pressure involved, and thus may possibly set a limit to economical height. Furthermore, in plants which have no system of maintaining a uniform charge, any trouble with poor slag at the spout is more serious with a high column than with a low one, because the high column does not allow corrective charges to descend so quickly to the breast, where they are needed. It also makes the handling of deep crusts more difficult. On the other hand, a higher column prolongs the period of gradual heating in the upper part of the furnace, and reduces the waste of heat in flue gases. Furthermore, it tends to prevent the escape of unconsumed oxygen, by exposing the charge for a longer time to oxidation. The column must be at least high enough to insure an even distribution

of blast in the lower part of the furnace, and thus depends to some extent on the width at the tuyeres.

For any one length of furnace, an increase in width at the tuyeres means an increase in hearth area. If the volume of blast per square foot of hearth is to remain unchanged, either the blast pressure must be raised or the tuyere area increased. The latter course, which is the more economical, will sooner or later bring up some interesting problems in jacket design. To some extent, the width at the tuyeres influences the width on top, to form the desired bosh. As the width on top increases, the difficulties of side charging from cars, as practiced at Great Falls, will probably call for some system of overhead charging, and thus introduce new mechanical features. Although systems of this sort have been worked out at other plants, the unusual widths which seem possible in future furnaces may bring up problems of their own.

Finally, the oxygen of the blast is a part of the furnace charge, and the method of blast supply should allow some sort of control over the volume, and hence the weight, of oxygen received by the furnace.

TABLE I.—*Details of Operation of the Blast-Furnace Department during August, 1894.*

Tonnages.

Blast Furnace No.	No. of Charges Smelted.	Total Tons Smelted, Exclusive Fuel.	Average Pounds Weight per Charge, Exclusive Fuel.	Pounds Fuel.		Per Cent. of Fuel.
				Coke.	Anthracite Coal.	
1	3,938	3,938.0	2,000	82,817.0	31,370	10.9
2	3,088	3,128.0	2,026	70,464.0	. . .	11.3
3	3,032	3,025.0	1,995	68,992.0	. .	11.4
4	1,670	1,671.0	2,001	38,468.0	. . .	11.5

Furnaces Nos. 1, 2, and 3 remained in operation throughout the month. No. 4 was blown in on the 14th.

The flue dust was mixed with water to form a paste or mud, and charged wet into the furnaces.

Average Composition of Charge.

Blast Furnace No.	Per Cent. of								
	Lime Rock.	Converter Slag.	Blast Furnace Slag and Chips.	Furnace Refuse Slag.	Concentrates.	First-Class Ore.	Flue Dust.	Matte.	Calcine.
1	0.23	40.50	1.78	0.04	0.49	0.28	0.47	56.62	0.02
2	13.17	31.39	1.62	3.33	24.58	13.74	0.84	7.08	4.61
3	14.71	27.38	1.70	3.56	28.15	15.18	. .	3 33	5.14
4	15.58	25.50	1.22	3.96	28.23	16.61	. .	0.51	8.38

TABLE II.—*Results of a Test on the Allen Process of Bessemerization in the Blast-Furnace Crucible, April, 1895.*

All Tests run on Blast Furnace No. 3.

Test run on April 9. [a]

Time Sampled.	Matte Grade.	Corresponding Slag					
	Per Cent. Cu.	Per Cent. Cu.	Per Cent. SiO_2.	Per Cent. Al_2O_3	Per Cent. FeO.	Per Cent. CaO.	Oxygen Ratio.
Before Bessemerizing..	39.6	. . .	. . . .	. . . .	. . . .	. . .	. . . .
After Bessemerizing....	51.5	. . .	. . . .	. . . .	. . . .	. . .	. . . .

Test run on April 11.

Before Bessemerizing .	37.5	. . .	. . . .	. . . .	. . . .	. . . .	. . . .
After blowing 20 min.	38.0	0.3	35.6	9.9	37.0	16.7	1.8099
After blowing 1 hr. 40 m	45.3	1.0	33.8	8.2	38.7	17.5	1.6000
After blowing 2 hr.....	45.7	1.8	. . . .	. . . .	. . . .	. . . .	. . . .

Test run on April 13.

| Before Bessemerizing... | 39.0 | . . . | . . . | . . . . | . . . . | . . . . | . . . . |
| After blowing 1 hr. 30 m. | 39.3 | . . . | 38.0 | 9.4 | 29.3 | 21.3 | 1.9492 |

a. The composition of the charge on this date was as follows:

	Pounds.
Concentrates, ...	800
Ore, ...	350
Slag, ...	350
Lime rock, ...	300
Calcines, ...	200
Total, ...	2,000

TABLE III.—*Details of Operation of the Blast-Furnace Department during 1895–96.*

Year.	Month.	Blast Furnace No.	Composition of Charge. Tons. Total, Exclusive Fuel.	Ore, Concentrates, Calcine.	Matte.	Slags.	Lime Rock.	Composition of Charge. Per Cent. Ore, Concentrates, Cal.	Matte.	Slags.	Lime Rock.	24-Hour Days Operated.	Tons per Day, Exclusive Fuel.	Coke. Tons.	Per Cent.	Tons per Day per Sq. Ft. of Hearth Area.	Ounces Blast Pressure.
1895	July	1	4,227	216	2,923	1,022	66	5.1	69.2	24.2	1.5	30	140 9	412	9 7	5.75	4.8
		2	2,816	1,402	466	549	399	49.9	16.6	19 2	14.3	25	112.6	312	11.1	4.60	8.8
		3	3,764	2,189	42	870	663	58.2	1 1	23.1	17.6	28½	132.1	406	10.8	4.40	8.8
		4	2,002	1,218	39	365	380	60.9	1 9	18.2	19.0	18½	108.2	222	11.1	5.09	8.6
	Aug.	3	3,911	2,281	99	789	742	58.3	2.5	20.2	19.0	31	126.2	434	11.1	4.21	...
1896	Apr.	3	4,208	2,707	. ...	621	880	64 3	......	14.8	20.9	27½	153 0	425	10.1	5.10	.....
	May	3	4,475	2,778	... .	645	1,052	62.1	......	14.4	23.5	30	149.2	450	10.1	4.97	.....
	Jun.	3	3,306	2,003	... .	550	758	60.6	......	16.6	22.8	25½	129.6	335	10 1	4.82	.....
	July	1	1,951	513	573	673	192	26.3	29.3	34.5	9.9	15	130 1	188	9.5	5.31	9.6
		2	3,429	1,237	794	946	452	36.0	23.2	27.6	13.2	26½	129 4	332	9 7	5.28	13,0
		3	2,843	1,756	10	425	652	61.7	0 4	14.9	23.0	19	149.6	285	10 0	4.99	15.8
		4	3,467	2,039	149	525	754	58.8	4.3	15.1	21.8	28½	121.6	348	10 0	4.72	12.8
	Aug.	1	3,843	1,192	982	1,292	377	31.0	25.6	33.6	9.8	30½	125 3	365	9.5	5.11	......
		2	2,798	1,680	147	451	564	58.3	5.2	16.4	20.1	26½	104.9	281	10 0	4.28	.....
		3	3,637	2,211	....	553	873	60.8	.....	15.2	24.0	24½	147.4	366	10.1	4.91	......
		4	3,405	2,082	11	507	805	61.2	0 3	14.9	23.6	29½	114.8	342	10.0	5.40	......
	Sept.	1	3,845	615	1,548	1,545	137	15.9	40.3	40.8	3.5	29	132 6	358	9.2	5.41	......
		2	2,123	1,143	38	521	421	53 8	1.8	24.5	19 9	18½	115.8	213	10 0	4.73	......
		3	4,436	2,568	18	764	1,086	58.0	0 4	17.2	24.4	28½	154.7	443	10 0	5.16	......
		4	3,288	1,904	10	571	803	57.9	0.3	17.4	24 4	28½	116 0	329	10.0	5.46	.... .
	Oct.	1	5,131	570	3,028	1,480	53	11.1	58.9	28.9	1.1	31	165 5	463	9.0	6.76	......
		2	3,943	1,976	239	959	770	50.2	6.0	24.3	19 5	31	127.2	394	10.0	5.19	... ...
		3	4,544	2,318	......	1,263	963	51.0	......	27.8	21.2	29	156.7	455	10.0	5.22	.....
		4	3,867	1,979	.	1,064	824	51.2	.. ..	27.5	21.3	31	124.7	387	10.0	5.87	. .....

TABLE IV.—*Details of Operation of Blast Furnace No. 1, July, 1897, to August, 1897, Inclusive.*

Dimensions of Furnace.—Hearth section, 42 by 180 in.   Height, center line of tuyeres to charging floor, 24 ft. 5 in.   Section at top of upper jackets, 48 by 180 in.

The average weight of charge is not recorded, but is given in another summary for February, 1898, as 2,000 lb.   All the charges here recorded were probably of about this weight.   On this assumption, the percentage of coke varied from 7.0 to 10.0 per cent.

The average number of charges per operating day, for the entire period covered by this tabulation, was 318.7.

Year.	Month.	Number of Charges Smelted per Day.	Grade of Matte, Per Cent. Cu.	Slag. Per Cent. SiO₂.	Per Cent. FeO.	Per Cent. Al₂O₃.	Per Cent. CaO.	Oxygen Ratio.	Pounds Coke per Charge.	Number of Days Operated.	Ounces Blast Pressure.
1897.	July	229	47.5	40.2	30.4	12.5	13.2	2.5827	199.6	9½	35.3
	Aug.	241.5	44.4	41.2	26.7	10.3	17.9	2.4217	200.0	17	34.0
	Sept.	266.7	47.4	40.5	27.1	11.9	18.2	2.4109	200.0	18	24.2
	Oct.	323.6	47.6	39.5	25.3	....	20.0	....	192.0	23	40.7
	Nov.	351.3	45.3	39.8	29.5	12.2	14.6	2.3019	173.4	18¼	32.0
	Dec.	340.4	52.4	40.4	31.5	11.2	17.0	2.2479	152.0	27	24.3
1898.	Jan.	331.7	53.0	40.0	28.5	9.8	17.9	2 2524	141 8	29¾	27.7
	Feb.	378.2	56.2	40.7	28.1	10.0	17.9	2.3116	140.6	25¼	33.3
	Mar.	347.8	57.2	40.2	28.6	8.9	17.8	2.2273	158.6	23½	29.8
	Apr.	300 0	57.8	40.1	27.2	8.9	20.3	2.1478	167.6	25	29.7
	May	326.0	58.7	39.0	33.3	10.5	15.8	2.1468	162.8	21	27.1
	June	314.9	56.0	39.8	31.2	10.2	14.4	2.3420	186.4	30	32.1
	July	299	56.2	42.7	25.5	9.8	18.8	2.4678	175.4	27¼	35.0
	Aug.	273	....	....	....	....	....	....	190.2	....	30.6

TABLE V.—*Details of Operation of Blast Furnace No. 1, September, October, November, December, 1898.*

All weights are Blast-Furnace Weights.

Furnace.	No. 1. 42 by 180 in.	24 ft. 5 in. high.		
Months, 1898.	September.	October.	November.	December.
Total No. of Charges smelted,	6,492	6,802	6,828	5,293
Coke, ... Tons.	649.1	743.0	758.6	573.8
Lime rock,	1,587.2	1,613.8	1,758.2	1,300.2
Conv. slag,	1,785.2	1,727.4	1,830.8	1,662.9
Concentrates,	273.1	536.8	604.1	782.2
First-class ore,	2,885.8	2,749.6	2,702.4	1,957.4
Moose ore,				
B. & M. matte,	60.7	334.0	229.5	195.0
Colorado matte,				
Refinery slag,	58.4	72.8	84.1	92.4
B. F. slag and chips,			9.5	1.6
Total tonnage, exclusive fuel,	6,650.4	7,034.4	7,218.6	5,991.7
Total tonnage, exclusive fuel and flux,	5,063.1	5,420.6	5,460.4	4,691.5
Number of days operated,	24¼	25	25½	20½
Tons per operating day,	274.2	281.4	283.1	292.3
Tons per operating day per square foot hearth,	5.22	5.36	5.39	5.57
Per cent. of coke,	9.8	10.6	10.5	9.6
Height of charge col. in feet,	13	15	15.1	15.9
Grade of matte, per cent. Cu,	54.6	54.5	55.1	51.5
Per cent. SiO_2,	41.2	41.8	41.2	40.5
Per cent. FeO,	25.4	23.4	23.5	24.3
Per cent. Al_2O_3,	9.2	8.4	8.4	9.6
Per cent. CaO,	20.5	22.0	22.8	22.0
Oxygen ratio (base $=1$, acid $=$),	2.2739	2.2735	2.1978	2.2235
Blast pressure, Oz,	24.7	24.5	27.5	22.2

(*Composition of Charge* brackets the rows from Coke through B. F. slag and chips; *Slag* brackets the Per cent. rows.)

TABLE VI.—*Details of Operation of Blast Furnaces Nos. 3 and 4, September, October, November, December, 1898.*

All Weights are Blast-Furnace Weights.

Furnace.	Blast Furnace No. 3. 96 by 120 in.				Blast Furnace No. 4. 34 by 40 in			
Month, 1898.	Sept.	Oct.	Nov.	Dec.	Sept.	Oct.	Nov.	Dec.
Total number of charges smelted,	3,943	4,116	4,523	4,276	3,717	3,333	3,751	3,749
Coke, ... Tons.	396.0	444.8	444 3	434 3	372.4	368.3	376 5	383.4
Lime rock,	129 0	529.8	516.7	280.3	80.9	822.5	422.1	225.6
Conv. slag,	1,600.7	1,322.8	1,108.0	1,593.5	1,341 3	1,089.7	937.8	1,354.0
Concentrates,	1,251.2	1,389.0	1,237 6	1,606.8	942.1	1,218.4	1,000.7	1,306.4
First-class ore,	155 2	855.0	449.8		80.5	469.7	359.1	
Moose ore,		0 5	221.0	188.9		0.3	188.0	167.0
B. & M. matte,	757.7	237.7	572.1	561,1	1,150.9	373 4	558 0	622.3
Colorado matte,	156 2	2.0	466.0	326.0	204 1	2 4	424 9	312.9
Refinery slag,		1.4						
B. F. slag and chips,			10.0	7.5			7.4	6.7
Total tonnage, exclusive fuel,	4,050 0	4,338.2	4,581.2	5,564.6	3,799.8	3,475.4	3,898.0	3,994.9
Total tonnage, excl. fuel and flux,	3,921.0	3,808.4	4,064.5	4,283.8	3,718 8	3,153.9	3,475.9	3,769.3
Number of days operated,	27¼	29½	27	28	26½	29½	30	30
Tons per operating day,	148 6	147.1	169.7	163.0	143.4	117.8	129.9	133.2
Tons per op. day per sq. ft. hearth,	4.95	4.90	5.66	5.43	6.75	5.54	6.11	6.27
Per cent. of coke,	9.8	10.3	9.7	9.5	9.8	10.6	9.7	9.6
Grade of matte, per cent. Cu,	51 8	53.0	53.9	51.0	52.1	51.5	55.1	51.6
Per cent. SiO_2,	39.0	41.4	42.2	40.2	37.3	41.5	41.5	39.6
Per cent FeO,	41.8	33.3	30.6	36.8	44.8	34.3	31.3	38.2
Per cent. Al_2O_3,	10 1	8.6	9.4	10.1	10.3	8 9	9 5	10.1
Per cent. CaO,	4 9	12.7	13.1	11.0	4.7	11.8	13.4	11.0
Oxygen ratio (base $=1$, acid $=$)..	2.3738	2.3545	2.5374	2.2988	2.1964	2.3791	2.4513	2.2096
Blast pressure, Oz.,	16.7	18.6	17.2	14.4	12.8	13.7	12.6	10.9

TABLE VII.—*Details of Operation of Blast Furnace No. 3 (72 by 180 in.) for the Entire Period of its Use.*

All Weights are Blast-Furnace Weights.

Months, 1899.		April.	May.	June.	July.	August	Sept.	Oct.	Nov.	Dec.
Total number of charges		5,242	10,557	1,743	11,787	13,464	11,714	8,100	9,560	8,180
Coke. Tons.		449.2	992.1	187.8	1,093.9	1,283.8	1,109.0	801.4	969.3	9,100
Lime rock.		1,272.8	2,525.5	447.2	2,876.3	3,007.4	2,764.2	1,728.3	2,491.1	2,036.3
Converter slag		1,282.5	3,021.1	559.9	3,135.7	3,398.2	2,512.3	1,727.0	2,107.2	1,852.6
First-class ore		1,397.4	2,387.1	318.5	3,879.1	4,128.8	3,979.9	2,285.8	3,366.4	2,450.5
Moose ore		19.0	115.2	....	306.7	207.6	....	.. ..		78.3
Coarse concentrates ..		1,340.5	2,988.5	585.2	1,891 4	3,020.2	2,428.1	2,106.3	1,765.5	1,922.5
B. and M. matte ..		....	....	....	....	....	....	194.0	.50.0	24.0
Refinery slag		32.0	62.8	10.8	....	....	....	....	....	....
Calcine lump........		2.1	3.4	....	....	....	....	. ... ..	25.5	10.6
Mud a		... .	....	....	....	....	....	8.7	20.0	....
Reverb chips........		....	....	....	....	18.8	115.4	....	....	. ... ..
Total tonnage, exclusive fuel		5,346.3	11,103.6	1,771.6	12,089.2	13,790.0	11,799.9	8,050.1	9,825.7	8,379.8
Total tonn., excl. fuel and flux		4,073.5	8,578.1	1,324.4	9,212.0	10,782.6	9,035.7	6,321.8	7,334.6	6,343.5
Number of days operated.... ..		14	31	7	28¾	31	30	24¾	29½	30
Tons per operating day... . .		381.9	358.2	253.1	420.5	444.8	393.3	325.3	333.1	279.3
Tons per op. day per sq. ft. hearth........		4.24	39 8	2.81	4.67	4.94	4.37	3.61	3.70	3.10
Per cent. of coke		8.4	8.9	10.6	9.0	9.3	9.4	10.0	9.9	10.9
Height of chg. col. ab. tuyers		12.7 ft.	13.2 ft.	13.4 ft.	14 ft.5	14 ft.5	14.3 ft.5	No Rec.	No Rec.	No Rec.
Grade of matte, per cent Cu.		48.6	48.4	53.0	51.1	48.8	49.5	49 6	52.7	54.2
Slag.	Per cent. SiO₂	42.9	42.7	40.8	41.9	42.7	41.9	42.6	42.1	42.3
	Per cent. FeO.	21.0	22.0	23.8	23.2	24.5	25.8	27.3	24.1	25.8
	Per cent. Al₂O₃	10.1	10.1	10.0	10.0	10.1	10.0	10.1	10.1	10.1
	Per cent. CaO	22.4	22.3	22.8	22.1	20.3	19.8	18.0	22.5	21.1
Oxygen ratio (base=1, acid=)		2.4887	2.4298	2.2305	2 3473	2.4342	2.3617	2.4384	2.2954	2.3305
Blast pressure, Oz............... .		28	35	34	30	36	34	31	31.6	33.3

a Probably flue dust, specially prepared for blast-furnace treatment.
Record incomplete.

TABLE VIII.—*Details of Operation of Blast Furnace No. 3 (44 in. by 180 in.), January, February, March, 1900.*

All Weights are Blast-Furnace Weights.

Months, 1900.		January.	February.	March.
Total No. of charges smelted, ...		4,540	6,410	8,504.0
Tons coal, ...		48.7	46.7	74.0
Tons coke, ...		384.5	602.1	786.8
Composition of Charge.	Lime rock, ... Tons	1,209.5	1,223.0	2,191.5
	Converter slag.	932.3	1,816.6	1,816.2
	Coarse concentrates, ...	985.1	1,844.9	1,766.7
	First-class ore, ...	1,496.7	1,550.7	2,830.9
	Moose ore, ...	7.0	24.5	71.3
	B. and M. matte, ...	21.7	.. .. .	39.0
	Refinery slag, ...	2.2	.. .. .	.. .. .
	Calcine lumps, ...	.. .. .	.. .. .	3 5
Total tonnage, exclusive fuel, ...		4,654.5	6,459.7	8,719.1
Total tonnage, exclusive fuel and flux, ...		3,445.0	5,236.7	6,527.6
Number of days operated, ...		14	28	30¼
Tons per operating day, ...		332.5	230.7	283.5
Tons per operating day per sq. ft. hearth, . . .		6.05	4.19	5.15
Per cent. of fuel, ...		9.3	10.0	99
Blast pressure, ounces, ...		29.6	33.4	56
Grade of matte, per cent. Cu, ...		52.6	51.1	493
Slag.	Per cent. SiO₂	40.8	41.4	42.2
	Per cent. FeO	22.9	27.7	21.7
	Per cent. Al₂O₃.	9.7	9.8	9.9
	Per cent. CaO	22.3	17.7	20.9
	Oxygen ratio (base = 1; acid =), . . .	2.2345	2.3666	2.5032

TABLE IX.—*Details of Operation of Blast Furnace No. 2 (56 by 180 in.) for each quarter-year, February, 1899, to September, 1900, inclusive.*

All Weights are Blast-Furnace Weights.

Period.	1899 February-March.	1899 April-June.	1899 July-September	1899 October-December.	1900 January-March	1900 April-June.	1900 July-September.
Total tons coke, ...	1,434.6	2,764.0	2,457.5	2,775.4	2.679.1	2,064.1	2,768.4
Total tons coal,.........	.. ..	.... ..	... .	....	219 4	186 9	....
Per cent. fuel,.	8 0	8.3	9 3	9.5	9 8	9.2	8.4
Composition of Charge. Total tons lime rock	4,124.0	8,304.6	6,187.0	7,273.3	7,248 2	6,014.8	8,073.3
Total tons conv. slag........	4,887.5	7,993.2	6,245.6	5,904 2	6,413.2	4,585.2	7,196.7
Total tons first-class ore	4,868.5	8,744.9	8,610.8	9,562.1	9,176.5	7,470.4	8,878.6
Total tons second-class ore...	113.9	....	....	....	....	....	....
Total tons Moose ore	....	118.3	423.4	91.2	252.6	413.0	355.4
Total tons coarse concentrates.............. ...	4,122 9	8,096.2	4,819 0	6,082.5	6,554 3	4,770 0	7,141 3
Total tons B. & M matte.......	....	....	25.0	237.7	26 0	177 0	74.0
Total tons refinery slag...........	231.0	144.1	....	....	....	....	14.2
Total tons blast-furnace chips,.	32.1	....	....	....	....	....	. ....
Total tons reverb chips.............	....	....	94.1	....	. .. .	....	....
Total tons lump calcine	....	8.1	8.5	26.2	0.9	... .	....
Total tons refinery brick....	....	....	....	....	. ..	1 8	....
Total tons briquettes	....	....	....	....	. ..	523 1	1,379.8
Total tons mud a.	....	....	....	22 1	....	....	
Number of operating days	49¼	75¾	67½	83¾	82½	63	83
Tons per operating day, exclusive fuel	363.3	441.0	391 3	350 1	360 0	350 2	398.8
Tons per operating day, excl. fuel and flux .	279.6	331.4	299.7	262.8	272 1	284 8	301.6
Tons per operating day, per square ft. hearth	5.19	6.30	5 59	5.00	5 14	5 43	5 70
Total number of charges	17,197	32,038	25,989	28,657	28,991	23,287	31,868
Average weight per charge...	2,081	2,085	2,033	2,034	2,049	2,057	2,111
Blast pressure, oz	25.7	34.7	33.3	32 8	36 5	33.9	30.8
Height of charge column, feet	12.9	13.6	....	....	. .....	.. ..	....
Grade of matte, per cent	51.3	50.3	50.1	51.9	51 7	52.7	53.9
Slag. Per cent SiO2	40.4	52.4	42.3	42.4	41.6	42 2	41.5
Per cent. FeO.	24.2	21.1	23.8	24.8	23 7	21.1	22.4
Per cent. Al2O3......	10.0	10.1	10.1	10.1	10 0	9 5	8.7
Per cent. CaO........	21.9	23 6	21.4	21.5	20 8	22 8	22 9
Oxygen ratio (base = 1, acid =)....	2.2442	2.3797	2.3848	2.3848	2.3875	2.3946	2 2659

a Probably flue dust, specially prepared for blast-furnace treatment.

Thermal Effect of Blast-Furnace Jackets.

BY ROBERT P. ROBERTS, GREAT FALLS, MONT.

In order to obtain data on the thermal effect of the blast-furnace jacket and on the water consumption in these jackets a series of tests were run on the 56 by 180 in. blast furnaces at the Great Falls Reduction Department of the Anaconda Copper Mining Co., at Great Falls, Mont.

The arrangement of the jacket is shown in the transverse section of one of the furnaces (Fig. 1) on which the tests were made. The nose, spout, and breast jackets have independent water supplies and discharges. The end and side jackets are connected in pairs, as regards water supply, the feed water entering at the bottom of the lower jacket and overflowing into the bottom of the jacket immediately above it, and discharging into an independent overflow pipe for each pair of jackets.

Tests were run on the main side and end jackets, the breast jacket, the spout, and the nose. Measurements were taken of the quantity of water discharged per minute and the rise in temperature of the water in its passage through the jacket. The product of these two results, expressed in the proper units, evidently gives the number of units of heat removed per minute by the water. Sufficient work was done on each type of jacket to determine its behavior under different conditions, and the results from each set of conditions were checked by repeating the work from three to six times.

The method used was as follows: The overflow from the jacket to be tested was arranged to discharge at will into a barrel, mounted upon platform scales. The discharge temperature was read just before turning the water into the barrel, and just before turning it back to the sluice; if the test lasted more than a minute, additional temperature readings were taken at 1-min. intervals from start to finish. The elapsed time of running the water into the barrel was determined with a stop-watch, and this figure, combined with the in-

crease in weight of the barrel, gave the pounds of water discharged per minute.   The initial temperature of the water was measured with a thermometer immersed in the supply tank on the feed floor.

For the breast jacket, spout, and nose, a length of flexible hose was attached to the end of the overflow pipe, by which means the water could be turned into the barrel quickly and easily.   For the side and end jackets a more elaborate apparatus was necessary, on account of the large size of the overflow pipes.   Iron piping was used for this purpose, with combinations of elbows and loose joints, so arranged
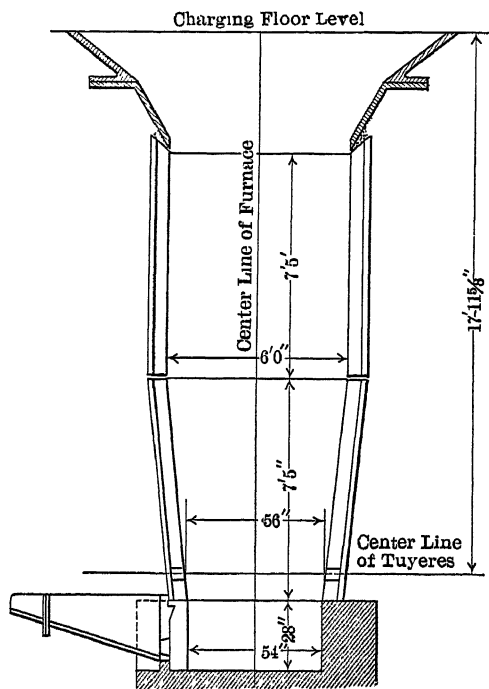
FIG. 1.—TRANSVERSE VERTICAL SECTION OF 56 BY 180 IN. BLAST FURNACE.

that the lower half length of the pipe could be swung horizontally from the sluice to the barrel, and back.   Both piping and hose were pierced, near their junctions with the joint overflows, to allow the introduction of the thermometer for measuring the discharge temperature.   The conditions were varied by changing the consumption of water, so that the period of time covered by one test varied considerably, being less than a minute in the case of the main side and end jackets with valves wide open, and as much as 19 min. for the breast jacket with the water reduced as far as safety allowed.

The following data will show the precision with which the work was done: Capacity of platform scales, 2,500 lb., reading to 0.5 lb.; these scales were checked with standard weights before beginning the test and found to be correct. The stop-watch read to 0.2 sec. Three thermometers were used altogether, one of them being broken in the course of the work. To explain the corrections applied to the temperature readings in the tabulations of results, the following description is given:

Thermometer A: Centigrade; graduated to 200° C. and reading to 1.0°.

Thermometer B: Fahrenheit; graduated to 600° F. and reading to 2.0°.

Thermometer C: Centigrade; graduated to 200° C. and reading to 1.0°.

Thermometers A and B checked exactly, for the range of temperatures required in the test. Thermometer A read 0.3° C. lower than thermometer C. In all the tests on the nose and the breast jacket, thermometer A was used at the supply tank, and B was used to read the discharge temperature. No correction is made in the temperatures for any of these tests. Thermometer B was broken just after running the sixth test on the spout; previous to this, the tests on the spout were run with the same arrangement of thermometers as in the case of the nose and the breast jacket. The seventh, eighth, and ninth tests of the spout were run with one thermometer, A doing duty both at the supply tank and the jacket discharge, since the temperature at the tank was found to remain practically constant. No correction is therefore necessary on any of the spout readings. All the tests on the main end jackets and the first 12 tests on the main side jackets were run with thermometer A at the tank and C at the jacket discharge. The discharge readings for these tests are therefore corrected by reducing them 0.3° C., to obtain the net rise in temperature of the water in the jackets. The last 36 tests on the main side jackets were run with the thermometer C at the tank and A at the jacket discharge. The discharge readings are therefore increased 0.3° C. in these tests. The amount of the correction is so small that it is of no importance whether the initial or the discharge temperature is the one to be corrected; in either case, the difference, or net rise, in temperature is not affected.

In the case of the main end and side jackets, each test was run on a set of two jackets, one upper and one lower, since there is an overflow for each set, but not for each jacket.

In the tabulations attached to the present report notes have been

added of the less important conditions affecting the results, such as dumping of charges, changes in the amount and temperature of slag, etc. However, a more detailed description of conditions is necessary, in order to fully explain the results; especially as it was found that these outside factors sometimes caused conflicting readings.

In the report on the tests of Nov. 3, 1910, a paragraph from Kent's *Steam Boiler Economy* was quoted to show that the rate of heat transmission of a clean metal plate, with a fire on one side and water on the other, depends on the difference of temperature on the two sides of the plate. Also, that this relation is disturbed by any accumulation on the plate of material which is a poor conductor of heat. In steam boilers, such an accumulation is furnished by scale on the inner, or water, side; while in a furnace jacket it is represented by crusts on the outer, or fire, side.

The thickness of the crust in the present test was found to affect the heat transmission to a considerable extent; although the interior temperature of the furnace, and the discharge temperature of the water, also had their effect. Usually, the readings were influenced by more than one of these factors, and different types of jacket showed the effect of each factor in different degree; so that the results can be understood only when all the test conditions are known.

Nose.—Beginning with the simplest case, that of the nose, the source of heat is a stream of liquid slag at a temperature not very far above its freezing point. In consequence, crusts are rapidly formed and are frequently barred out, so that the conducting power of the jacket varies from that of clean metal to that of a slag crust over an inch thick. Slag being a poor heat conductor, compared with the metals, we can expect a good deal of variation in the number of heat units carried away by the water.

In the first test, a moderate amount of crust was present before the readings began, and this slowly increased while they were being taken. It will be seen from the tabulation that after the fourth minute the temperature readings gradually dropped from 100° F. to 90° F. In the second test, all the crust was removed just previous to taking the first reading, but no barring was done after the test began. The temperature of the water rose for 2 min. and then decreased for 10 min., finally becoming constant as the crust reached a thickness so great that the heat transmission became too small to chill the slag below its freezing point. The remaining four tests were run with a normal accumulation of crust, and it is evident that, with one exception, the temperature of discharge gradually decreased while the readings were being taken. The exception occurs in the

fourth test, when a large amount of matte in the stream, and a large volume of slag, due to dumping fresh charges in the furnace, caused a rising temperature. Charges were also dumped during the second test, and may have retarded the downward trend of the temperature readings for a minute or two, without altogether preventing it. The curves are shown in Fig. 2.

Arrows indicate time of dumping charges.
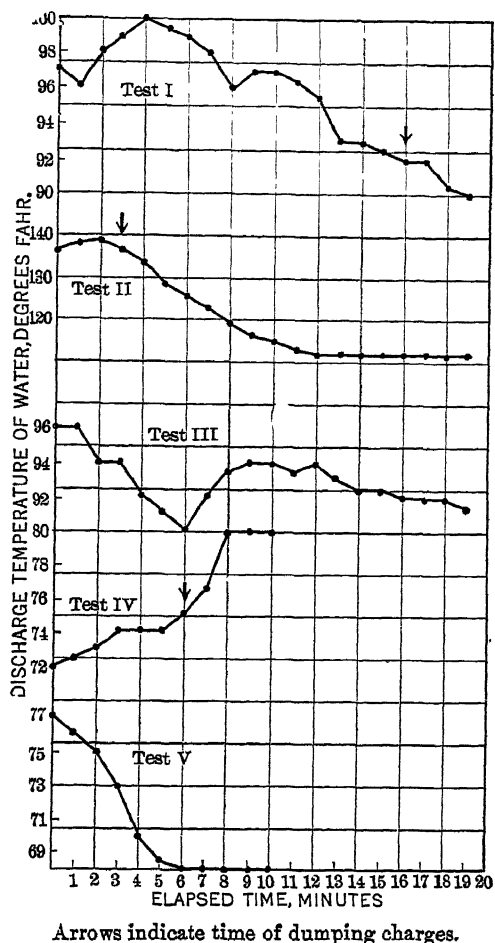
FIG. 2.—NOSE. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS I. TO V., INCLUSIVE.

The above discussion of temperature applies equally to the heat units removed by the water per minute, because the measurements of the discharge show that it remained practically constant unless altered by adjusting the valves. Changes in the thickness of the crust were

so rapid that they affected each individual test, and consequently each series, a series being a group of tests in which the flow of jacket water was kept constant. Now, comparing individual tests, the fifth and sixth tests were run with a reduced volume of slag flowing from the furnace, and the average discharge temperature in these two tests is lower than in the fourth test, which was run under the same conditions as to flow of water. In all three of these tests the amount of crust is recorded as normal, and it seems probable that the lower temperature of discharge in Tests V. and VI. is due to the reduced amount of slag. That this should be expected may be seen from the formula for the rate of heat transmission given by Kent, in which the temperature on the "fire" side of the transmitting plate is one of the factors. An increase in the volume of the slag stream covers a greater surface of the jacket with hot slag, and thus raises the average temperature over the whole surface of the "fire" side.

Finally, comparing series to series, we find that the three tests with a smaller volume, and therefore a higher temperature, of water show a smaller heat loss, or smaller rate of heat transmission. This is to be expected from the other factor in Kent's formula, which is the temperature on the "water" side of the transmitting plate. Thus all three factors—thickness of crust and temperature on the two sides of the plate (or jacket)—have influenced these results, the thickness of crust being apparently the most important.

Spout.—The crust in the spout is rarely entirely removed, except in case of a shut-down or of a freeze-up. Usually it is left to build up or cut out, according to the temperature of the slag stream, as it comes from the furnace. Whence the influence of the crust upon the rate of heat transmission should be almost constant. Furthermore, since the crust is so thick, it should tend to mask the effect of the other factors, since its own influence is correspondingly important. Of these other factors, the temperature on the "fire" side of the jacket should have greater effect than the temperature of the water, since it is subject to greater variation. Just as in the case of the nose, the temperature of the "fire" side varies more with the volume than with the actual temperature of the slag, since an increase in volume causes a large increase in the area of jacket covered by hot slag, raising the average temperature of the "fire" side more quickly than a few degrees difference in actual slag temperature.

These conclusions are all justified by the results of the work. Within narrow limits, the quantity of heat removed per minute, or the rate of heat transmission, remained constant, showing the greater influence of the crust in retarding and fixing the rate of heat trans-

mission and in obscuring the effects of other factors. This is still better shown by the average of the series, tabulated in the summary.

However, the discharge temperature readings show that the temperature of the " fire " side also had its effect, and these conditions apply equally well to the rate of heat transmission, since the volume of water in each series remained very nearly constant. Charges were dumped into the furnace just before starting the first test, and caused the usual rush of slag from the furnace, the stream then falling off while the readings were being taken. The readings decrease rapidly
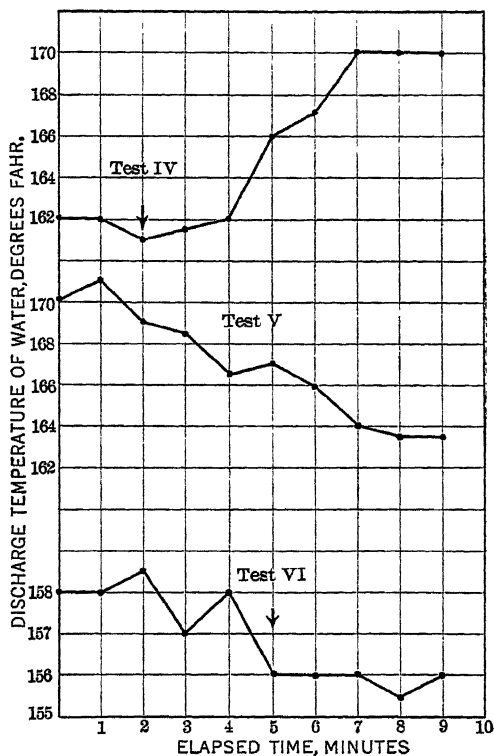
Arrows indicate time of dumping charges.

FIG. 3.—SPOUT. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS IV. TO VI., INCLUSIVE.

for a few minutes, and never return to the high temperature shown in the first reading. In the third test, charges were dumped soon after the start, and the temperature rose rapidly from that time. This is also the case with Test IV., as may be seen from the curve sheet, Fig. 3. Charges were dumped again just as the last test was begun, and the temperature again rose. In Test VI. the slag stream was very low towards the end, and the temperature gradually decreased.

The results do not show the effect of the water temperature. The first series, at medium volume and temperature of discharge, gives the lowest heat loss, while the second series, with the smallest volume and highest temperature of discharge, gives a result intermediate between the other two. The thickness of crust is again the most important factor, with the temperature of the " fire " side second.

Breast Jacket.—Although the breast jacket is part of the walls of the furnace, it usually differs from the main side and end jackets in being less subject to crusting. This is due to the constant flow of hot product past the breast jacket and out into the spout, and also to the omission of the two center tuyeres on the front of the furnace, which prevents any chilling from cold air at that point. The influence of crusting should therefore be less than with the nose and spout.

The results vary considerably, indicating that crusting actually is of minor importance. The first two series were run in the morning, when the slag stream was medium in both volume and temperature. The third series was run in the afternoon, and by that time the slag was hotter and flowing in greater quantity. Thus the conditions could not be kept uniform, and the effect of the water temperature is not properly shown by the results. However, the first two series, Tests I. to VI., inclusive, were run under nearly the same conditions, except as regards volume of water, and it will be seen that the hotter water carried away less heat than the colder, indicating the effect of the water temperature on the rate of heat transmission when other factors remain constant.

The highest heat loss is shown by the third series, when the temperature on the " fire " side of the jacket rose high enough to offset the rise in water temperature. Considering water temperature alone, the third series should give results intermediate between the other two, and the fact that it gives the highest results of all shows the greater importance of the fire temperature. This is also shown by the individual temperature readings. Although the breast jacket forms a part of the furnace wall, it also resembles the spout and nose in being in contact chiefly with the molten products, rather than with unsmelted or partly smelted charge materials. Therefore we should expect to find that the effect of dumping charges is to raise the fire temperature, by causing a larger flow of those products and thus bringing a greater number of heat units in contact with the jacket in a given time. Tests I., II., and VII. indicate that this is true, as can be seen from the curves, Fig. 4. Charges were also dumped in Tests III., VI., and IX., but without producing any effect on the temperatures. Of the remaining tests, Nos. IV. and V. show almost constant

temperatures, and No. VIII. a small rise after the first minute. No charges were dumped during these tests.

The controlling factor seems to be the furnace and slag temperature, with the water temperature of next importance.
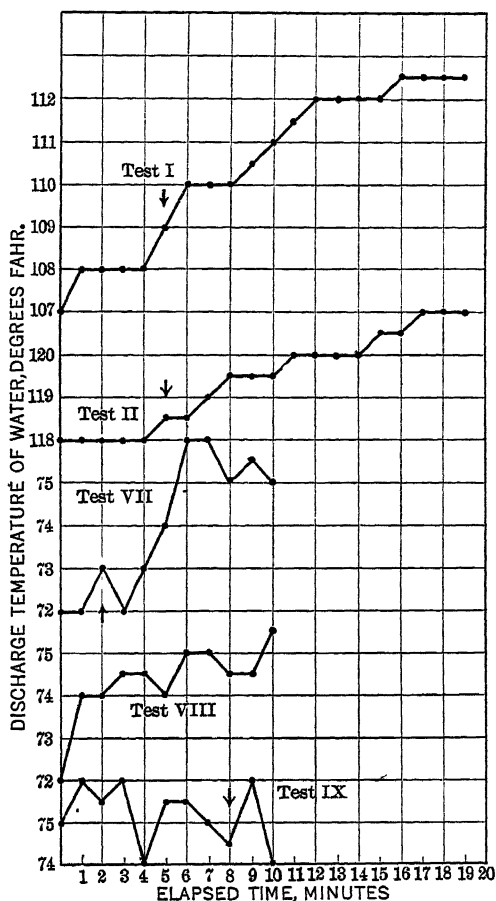
Test III. gives a straight line at constant temperature and is therefore omitted.

Arrows indicate time of dumping charges.

FIG. 4.—BREAST JACKET. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS I., II., VII., VIII., AND IX.

Main End Jackets.—The results on the main end jackets can be compared with similar data given in the report on the test of Nov. 3, 1910, in which the same method of work was followed and the same kind of conditions prevailed. Compared in this way, the present results are much lower than those of the previous test, though still close enough to serve as a general check.

Crust is nearly always present on the end jackets, even though it may not appear at the surface of the charge column, and it varies greatly in thickness. A loose, porous crust may allow cold blast from below to cool the upper portions of the jackets, while a dense, hard crust protects the jacket by poor conduction. The uncovered part of the jackets, above the charge column, may be exposed to higher temperatures than the portions covered by charge and more or less crusted.

The effect of the crusts in the present test is evident in the nearly constant value of the heat loss. The transmission of heat seems to have been retarded to such an extent that the effect of water temperature was of no importance. However, the fire (or furnace) temperature had greater influence, as may be seen from the last three tests. These were run at a time when the furnace was receiving charges in rapid succession, and being cooled down in the upper part of the charge column as a consequence. The heat loss in these three tests is the lowest of any, although all the other conditions remained about the same. Omitting these tests, the average B.t.u. removed per minute by the jacket during the last series (Tests XIII. to XVIII., inclusive) amounts to 9,583, which checks more closely with the 9,963 and 10,028 B.t.u. per minute of the other series. An inspection of the individual temperature readings shows the effect of dumping cold material against the jackets, and confirms the importance of the furnace temperature, for after each charge the water temperature dropped rapidly. This is well shown by the curves, Fig. 5, in which the water temperature is plotted against the regular time intervals of the readings.

Main Side Jackets.—The side and end jackets present very nearly the same conditions, the only difference being that the sides are more apt to be affected by blast spilling along the jackets, since the tuyeres are placed on the sides. Otherwise, the influence of crusts, of cold charge materials, and of general furnace temperature, should be the same for both types of jacket.

Tests were run on two different furnaces, Nos. 2 and 4. The heat loss in No. 2 varied from 2,762 to 16,752 B.t.u. per minute, and in No. 4 from 1,918 to 6,302 B.t.u. per minute. The average of the previous tests, November, 1910, gives a loss of 26,099 B.t.u. per minute, the furnace being No. 4. These results vary so widely that some explanation of them must be looked for in the condition of the furnace.

The first 12 tests of the present work were run on Dec. 3, 1910. Tests I. to VI. were run between 10:00 and 10:45 a.m. and

show a sudden drop in temperature in the last of the series, although the volume of water varied less than 1 per cent. through all the six tests. Tests VII. to XII. were run between 1:00 and 1:30 p.m. and show a sudden rise towards the end, followed by a smaller drop. The heat loss in Test XI. is about 23 per cent. greater than that in Test
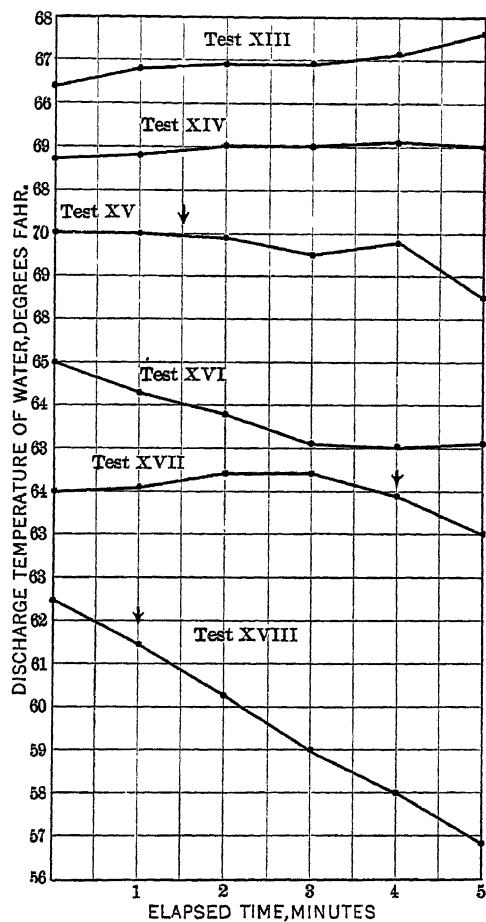
Arrows indicate time of dumping charges.

FIG. 5.—MAIN END JACKETS. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS XIII. TO XVIII., INCLUSIVE.

VII., due entirely to differences in temperature, since the volumes of water check almost exactly.

The fact that these changes in water temperature came near the end of the series in each case shows that they were not due to starting the tests too soon after changing the volume of water, or before

the new temperature had become constant. Trouble of this kind was encountered before, but in that case the temperatures varied during each separate test, and gradually became constant as the conditions adjusted themselves. The reverse is evidently true in the present results, and the variations can have been caused only by changing conditions within the furnace.

Tests XIII. to XXX., inclusive, were run on Dec. 14, 1910, on Furnace No. 2. The first six of these give a heat loss comparable with that found on Nov. 3, although much lower. They were run between 10:00 a.m. and noon, with a small amount of crust showing at the surface of the column. Tests XIX. and XXIV. were run between 1:00 and 3:00 p. m., and show only about 40 per cent. of the heat loss found in the first six, the volume of water being nearly the same. By this time a heavy, loose crust had developed, and the furnace was taking its blast poorly. The last six tests, XXV. to XXX., were run with a larger volume of water, and gave a heat loss checking fairly well with the second six. Finally, all the water possible was turned into the jacket, but the discharge temperature remained almost unchanged, and the work was transferred to Furnace No. 4.

The seventh, eighth, and ninth series, Tests XXXI. to XLVIII., were run on Dec. 15, 1910, on Furnace No. 4. The water was reduced on the northeast side jackets at 9:10 a.m. and temperature readings were then taken every 5 min. until no further rise was observed. These readings were as follows:

9:10 a.m., water reduced.	10:10 a.m., 36° C.
9:25 a.m., 16° C.	10:15 a.m., 33° C.
9:30 a.m., 19° C.	10:20 a.m., 33° C.
9:33 a.m, 22° C.	10:25 a.m., 35° C.
9:35 a.m., 25° C.	10:30 a.m., 36° C.
9:40 a.m., 27° C.	10:35 a.m., 37° C.
9:45 a.m., 28° C.	10:40 a.m., 34° C.
9:50 a.m., 31° C.	10:45 a.m., 35° C.
9:55 a.m., 33° C.	10:50 a.m., 36° C.
10:00 a.m., 37° C.	10:55 a.m., 36° C.
10:05 a.m., 38° C.	11:00 a m., 36° C.

These figures show that care was taken to obtain a constant temperature, since from 10:00 a.m. to 11:00 the readings were fairly uniform. The tests were then started, but during Test XXXIII. the temperature began to rise again, with the result that one of the later tests gave a heat loss nearly 30 per cent. higher than that in Test XXX. The last two series gave fairly uniform results, also checking the fourth and fifth series.

In reviewing all the results obtained on the side jackets no value

can be assigned to the heat loss. This is due to the variations in water temperature, occurring while the volume of water remained practically constant, and apparently following no law except that they were influenced by the interior conditions of the furnace. In one case the temperature of the water remained constant while the volume was doubled, and in many cases it varied over a wide range while the volume remained constant. There can be only one explanation of the changes: namely, the furnace temperature and the effect of crusts. Evidently the relation between heat loss and water temperature is much less important than the effect of these other factors, as can be seen in the summary. The greatest heat loss occurred in the third series, in which the water temperature was highest, while the lowest heat loss occurred in the sixth series, which had the second highest water temperature. The series that furnish the best check are the fourth, fifth, seventh, and eighth, with an average heat loss of 5,736 B.t.u. per minute. The water temperature in the eighth series is increased by about 300 per cent. in the fourth series, though the B.t.u. lost per minute check to within 7 per cent. This resembles the constant value of the heat loss found in the end jackets, and in both end and side jackets on Nov. 3. The remaining four series show the widest variation in heat loss, without any relation between this and the water temperature.

The curves, Fig. 6, show this temperature variation graphically and illustrate the regularity of the changes in certain tests, suggesting that a gradual change was going on at the same time within the furnace.

Effect of the Length of Furnace on Heat Loss Through Jackets.

A tabulation and a curve, Fig. 7, have been added, showing the percentage of area in the end jackets for different lengths of furnace. This percentage decreases as the furnace is lengthened, and the square feet of radiating surface (jacket area) per square foot of hearth also shows a decrease, though less in amount. A study of the curve shows that the greater part of the decrease is produced by lengthening the furnace from 15 to 60 ft. Beyond 60 ft. the saving in jacket area is very small.

The lengths of furnace at the Anaconda plant are 87 ft. and 51 ft., there being one of the former length and two of the latter. Calculating the proportion of end jacket area in the total, we find that in the 87-ft. furnace the end jackets furnish 6.1 per cent. of the whole area, and in the 51-ft. furnace, 9.9 per cent. The ratios of jacket area to hearth area in the furnaces are 6.78 and 7.07, respect-

ively.   These figures show that the 9 ft. of length added to the 51-ft. furnace to make the 60 ft. furnace produce a relatively greater saving than the 27 ft. of difference between the 60-ft. and the 87-ft. furnaces.

If reasonable figures can be deduced for the heat transmission of side and end jackets, from the tests now reported, these calculations
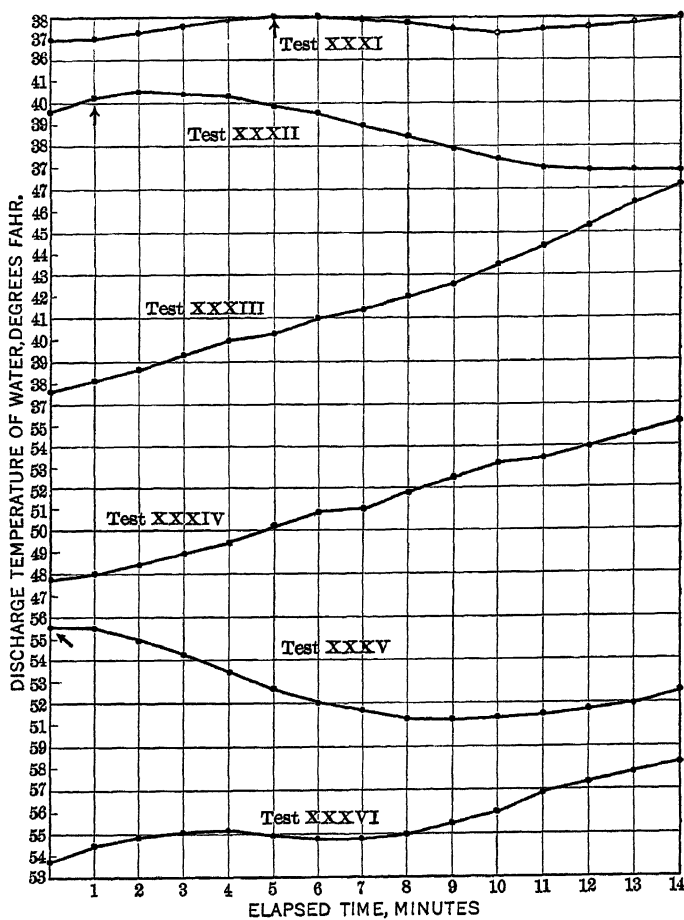
Arrows indicate time of dumping charges.

Fig. 6.—Main Side Jackets.   Variation in Temperature of Jacket
Discharge During Tests XXXI. to XXXVI., Inclusive.

of jacket area and saving in end jackets can be expressed in terms of heat loss.   A tabulation is presented, in which the most satisfactory results are averaged for the side jackets and all results are averaged for the end jackets, taking into account the work of Nov. 3, 1910. For this purpose, the side jacket tests of Dec. 3 are omitted, as the results are all very low, and the furnace was in such condition that

the work had to be discontinued before all the desired conditions could be tested. Omitting these, we find that the average heat loss of two side jackets (upper and lower) was about 8,400 B.t.u. per minute, and of eight side jackets, 33,600 B.t.u per minute. This figure is the unit in all subsequent calculations, as it represents the heat loss of all side jackets in the unit or 15-ft. furnace. For the end jackets, we find an average of 11,500 B.t.u. per minute with two jackets, and 23,000 B.t.u. per minute with four jackets, which represents the full end-jacket area of any length of furnace.
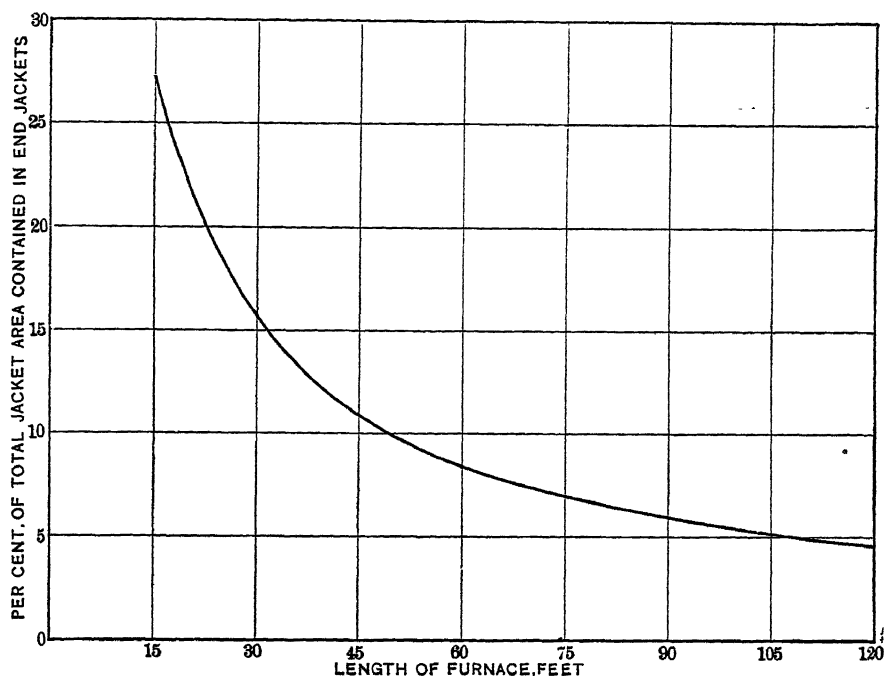
DATA.

Transverse Section : Width between upper jackets, 6 ft.
                    Width at tuyeres, 4 ft. 8 in.
Size of Upper Jackets : End, 7 ft. 5 in. by 6 ft. 6 in.
                        Side, 7 ft. 5 in. by 7 ft. 6 in.
Size of Lower Jackets : End, 7 ft. 5 in. high, 6 ft. 6 in. wide at top, 5 ft. wide at bottom.
                        Side, 7 ft. 5 in. vertical height, 7 ft. 5.45 in. slant height, 7 ft. 6 in. wide.
                        End jackets overlap side jackets 3 in. on each side.

FIG. 7.—CURVE SHOWING PER CENT. OF TOTAL BLAST-FURNACE JACKET AREA CONTAINED IN THE END JACKETS, FOR VARIOUS LENGTHS OF FURNACE.

The second part of the tabulation shows the heat loss of various furnaces, ranging in length from 15 to 120 ft., and including the two Anaconda types. Taking the heating value of blast-furnace coke for

December, 1910 (not ash-plant coke) as representing the average of our coke, these figures are converted into pounds of coke per minute. A column is also added showing the pounds of coke per minute equivalent to the heat loss of a number of 15-ft. furnaces whose total hearth area would equal the hearth area of the long, or combined, type. The latter figures are evidently greater than the actual coke equivalent of the heat loss for the combined furnaces and the ratio between the two may be expressed in percentage. This is done in the last column.

Test on the Thermal Effect of Blast-Furnace Jackets.
Nose.

Blast Furnace No. 2.   Nov. 29, 1910.

	Observed Data.			Calculated Data.				
Test No.	Net Wt of Water. Pounds.	Initial Temp. of Water. °C.	Elapsed Time. Min.-Sec.	Average Disch'g Temp. °F.	Rise in Temp. °F.	Pounds Water Per Minute.	Gallons Water Per Minute.	B.t.u. Removed Per Minute
I.	175.5	0.0	19–3.2	95.6	63.6	9.21	1.10	585.8
II.	166.0	0.0	19–0.8	120.8	88.8	8.73	1.05	775.2
III.	148 5	0.0	19–1.6	93.0	61.0	7.80	0.94	475.8
IV.	237.5	0.0	10–1.2	75.5	43.5	23.70	2.84	1031.0
V.	232.5	0.0	9–56.4	70.9	38.9	23.39	2.80	909.9
VI.	138.5	0.0	5–55.0	68.3	36.3	23.41	2.81	849.8

Detailed Temperature Readings and Notes.

Test I.		Test II.		Test III		Test IV.		Test V.		Test VI.	
Elapsed Time.	Disch'g Temp.	Elapsed Time	Disch'g Temp.	Elapsed Time.	Disch'g Temp.	Elapsed Time.	Disch'g Temp.	Elapsed Time.	Disch'g Temp	Elapsed Time.	Disch'g Temp.
Min. Sec.	°F		°F		°F		°F		°F		°F
0–0.0	97.0	0–0.0	136.0	0–0.0	96.0	0–0.0	72.0	0–0.0	77.0	0–0.0	69.5
1–0.0	96.0	1–0.0	137.5	1–0.0	96.0	1–0.0	72.5	1–0.0	76.0	1–0.0	68.5
2–0.0	98.0	2–0.0	138.0	2–0.0	94.0	2–0.0	73.0	2–0.0	75.0	2–0.0	68.0
3–0.0	99.0	3–0.0	136.0a	3–0.0	94.0	3–0.0	74.0	3–0.0	73.0	3–0.0	68.5
4–0.0	100.0	4–0.0	133.0	4–0.0	92.0	4–0.0	74.0	4–0.0	70.0	4–0.0	68.5
5–0.0	99.5	5–0.0	128.0	5–0.0	91.0	5–0.0	74.0	5–0.0	68.5	5–0.0	67.0
6–0.0	99.0	6–0.0	125.0	6–0.0	90.0	5–0.0	75.0a	6–0.0	68.0		
7–0.0	98.0	7–0.0	122.5	7–0.0	92.0	7–0.0	76.5b	7–0.0	68.0	Normal accumulation of crust.	
8–0.0	96.0	8–0.0	119.0	8–0.0	93.5	8–0.0	80.0	8–0.0	68.0	Amount of slag reduced as in Test V.	
9–0.0	97.0	9–0.0	116.0	9–0.0	94.0	9–0.0	80.0	9–0.0	68.0		
10–0.0	97.0	10–0.0	115.0	10–0.0	94.0	10–1.2	80.0	9–56.4	68.0		
11–0.0	96.5	11–0.0	113.0	11–0.0	93.5						
12–0.0	95.5	12–0.0	112.0	12–0.0	94.0	Normal accumulation of crust.		Normal accumulation of crust. Quantity of slag falling off.			
13–0.0	93.0	13–0.0	112.0	13–0.0	93.0						
14–0.0	93.0	14–0.0	112.0	14–0.0	92.5						
15–0.0	92.5	15–0.0	112.0	15–0.0	92.5						
16–0.0	92.0a	16–0.0	112.0	16–0.0	92.0						
17–0.0	92.0	17–0.0	112.0	17–0.0	92.0						
18–0.0	90.5	18–0.0	112.0	18–0.0	92.0						
19–3.2	90.0	19–0.8	112.0	19–1.6	91.5						
Considerable crust formed— none removed.		Crust removed just previous to test—then left untouched.		Normal accumulation of crust.							

a. Charges dumped.
b. Large amount of matte in stream.

If we use the term "fuel loss" to mean the pounds of coke required per minute to supply the heat loss through the jackets, it is evident that the fuel loss decreases rapidly between lengths of 15 and 60 ft., and very little beyond 60 ft. The 60-ft. furnace shows a fuel loss equal to 69.6 per cent. of the fuel loss in four 15-ft. furnaces, while the 120-ft. furnace shows a fuel loss of 64.5 per cent. of that in eight 15-ft. furnaces. Even doubling the length of our present furnaces would reduce the fuel loss about 20 per cent., and increasing it to 60 ft. would reduce it another 10 per cent. Beyond this there seems to be little chance of economy in heat, although other consideations might make still longer furnaces advisable.

Spout.

Blast Furnace No. 2. Dec. 1, 1910.

Test No.	Observed Data.					Calculated Data.		
	Net Weight of Water. Pounds.	Initial Temperature of Water.	Elapsed Time, Min.-Sec.	Average Discharging Temperature. °F.	Rise in Temperature. °F.	Pounds Water per Minute.	Gallons Water per Minute.	B.t.u. Removed per Minute.
I.	360.0	0.0 °C	5– 0.7	98.9	66.9	71.83	8.61	4,805
II.	362.0	0.0 °C	5– 3.6	90.7	58.7	71.54	8.58	4,199
III.	358.5	0.0 °C	5– 0.3	92.2	60.2	71.63	8.59	4,312
IV.	313.5	0.0 °C	9– 1.2	165.2	133.2	34.76	4.17	4,630
V.	311.5	0.0 °C	9– 1.3	166.9	134.9	34.53	4.14	4,658
VI.	309.5	0.0 °C	9– 1.2	156.9	124.9	34.31	4.11	4,285
VII.	420.0	0.0 °C	3– 0.7	67.4	35.4	139.46	16.72	4,937
VIII.	417.0	0.0 °C	2–59 6	65.8	33.8	139.31	16.70	4,709
IX.	419.5	0.0 °C	3– 0.6	64.9	32.9	139.37	16.71	4,585

Detailed Temperature Readings and Notes.

In Tests I. to VI., inclusive, discharging temperature read with Fahrenheit thermometer; Tests VII. to IX., Centigrade.

Test I.		Test II.		Test III.		Test IV.		Test V.		Test VI.		Test VII.		Test VIII.		Test IX.	
Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °F.	Elapsed Time.	Discharging Temperature. °C.	Elapsed Time.	Discharging Temperature. °C.
0–0.0	100.0	0–0.0	98.0	0–0.0	90.0	0–0.0	162.0	0–0.0	170.0	0–0.0	158.0	0–0.0	19.75	0–0.0	18.00	0–0.0	17.50a
1–0.0	99.5	1–0.0	92.5	1–0.0	90.0a	1–0.0	162.0	1–0.0	171.0	1–0.0	158.0	1–0.0	20.25	1–0.0	19.00	1–0.0	18.00
2–0.0	99.0	2–0.0	90.0	2–0.0	91.0	2–0.0	161.0a	2–0.0	169.0	2–0.0	158.5	2–0 0	19.25	2–0.0	19.00	2–0 0	18.00
3–0.0	97.5	3–0.0	90.0	3–0.0	92.0	3–0.0	161.5	3–0.0	168.5	3–0.0	157.0	3–0 7	19.50	2–59.6	19.00	3–0.6	19.50
4–0.0	98.5	4–0.0	89.5	4–0.0	94.0	4–0.0	162.0	4–0.0	166.5	4–0.0	158.0			Slag stream low			
5–0.7	99.0	5–3.6	89.0	5–0.3	96.0	5–0.0	166.0	5–0.0	167.0	5–0.0	156.0b						
Charges dumped just before test began		Volume of slag less than in Test I				6–0.0	167.0	6–0.0	166.0	6–0.0	156.0						
						7–0.0	170.0	7–0.0	164.0	7–0.0	156.0						
						8–0.0	170.0	8–0.0	163.5	8–0.0	155.5						
						9–1.2	170.0	9–1.3	163.5	9–1.2	156.0						
								Slag stream low									

a Charges dumped.
b Slag stream much reduced in volume.

Breast Jacket.

Blast Furnace No 2.   Nov. 30, 1910.

Test No.	Observed Data			Calculated Data				
	Net Wt. of Water Pounds.	Initial Temp. of Water.	Elapsed Time Min.–Sec	Average Disch'g Temp °F.	Rise in Temp. °F.	Pounds Water per Minute.	Gallons Water per Minute.	B.t u. Removed per Minute.
I	197.0	0.0°C	19–0.4	110.5	78.5	10.36	1.24	813
II.	184.5	0 0°C	19–4.6	119.4	87.4	9.67	1.16	845
III.	178.5	0 0°C	19–1.7	122.0	90.0	9.38	1.12	844
IV.	345.5	0.0°C	5–6.2	47.2	15.2	67.70	8.12	1,029
V.	340.5	0.0°C	5–2.6	47.6	15 6	67.51	8.09	1,053
VI.	339.0	0.0°C	5–1.0	48.5	16.5	67.57	8.10	1,115
VII.	339.0	0 0°C	10–3.6	74 0	42.0	33.70	4.04	1,415
VIII.	331.0	0.0°C	10–1.4	74.3	42.3	33.02	3.96	1,397
IX.	327.0	0.0°C	10–2.2	75.2	43.2	32.58	3.91	1,408

Detailed Temperature Readings and Notes.

Tests VII , VIII , and IX. were run in the afternoon, with a larger and hotter slag stream than in the morning when Tests I. to VI. were run.

Test I		Test II		Test III.		Test IV.		Test V		Test VI	
Elapsed Time.	Discharging Temperature °F	Elapsed Time.	Discharging Temperature °F.	Elapsed Time.	Discharging Temperature °F.	Elapsed Time.	Discharging Temperature °F.	Elapsed Time.	Discharging Temperature °F.	Elapsed Time	Discharging Temperature °F.
0–0.0	107.0	0-0.0	118.0	0–0.0	122.0	0–0.0	47.5	0–0.0	47.5	0–0.0	47.0
1–0.0	108.0	1-0.0	118.0	1–0.0	122 0	1–0.0	47.0	1–0.0	48 0	1–0.0	50.0
2–0.0	108.0	2-0.0	118.0	2–0.0	122.0	2–0.0	47.0	2–0.0	48 0	2–0.0	48.5a
3–0.0	108.0	3-0.0	118.0	3–0.0	122.0	3–0.0	47.5	3–0.0	48.0	3–0.0	49.0
4–0.0	108.0	4-0.0	118.0	4–0.0	122.0	4–0.0	47.0	4–0.0	48.0	4–0.0	48.5
5–0.0	109 0a	5-0.0	118.5a	5–0.0	122.0	5–6.2	47.0	5–2.6	46.0	5–1.0	48.0
6–0.0	110.0	6-0.0	118.5	6–0.0	122 0	Test VII.		Test VIII.		Test IX	
7–0.0	110.0	7-0.0	119 0	7–0.0	122.0a	0–0.0	72.0	0–0.0	72.0	0–0.0	75.0a
8–0.0	110.0	8-0.0	119.5	8–0.0	122.0	1–0.0	72.0	1–0.0	74.0	1–0.0	76.0
9–0.0	110.5	9-0.0	119.5	9–0.0	122 0	2–0.0	73.0a	2–0.0	74.0	2–0.0	75.5
10–0.0	111.0	10-0.0	119.5	10–0.0	122 0	3–0.0	72.0	3–0.0	74.5	3–0.0	76.0
11–0.0	111.5	11-0.0	120.0	11–0.0	122.0	4–0.0	73.0	4–0.0	74.5	4–0.0	74.0
12–0.0	112.0	12-0.0	120 0	12–0.0	122.0	5–0.0	74.0	5–0.0	74.0	5–0.0	75.5
13–0.0	112.0	13-0.0	120.0	13–0.0	122.0	6–0.0	76.0	6–0.0	75.0	6–0.0	75.5
14–0.0	112.0	14-0.0	120.0	14–0.0	122.0	7–0.0	76.0	7–0.0	75.0	7–0.0	75.0
15–0.0	112.0	15-0.0	120.5a	15–0.0	122.0	8–0.0	75.0	8–0.0	74.5	8–0.0	74.5a
16–0.0	112.5	16-0.0	120.5	16–0.0	122.0	9–0.0	75.5	9–0.0	74.5	9–0.0	76.0
17–0.0	112.5	17-0.0	121.0	17–0.0	122.0	10–3.6	75.0	10–1.4	75.5	10–2.2	74.0
18–0.0	112.5	18-0.0	121.0	18–0.0	122.0	....	....	....	....	....	....
19–0.4	112.5	19-4.6	121.0	19–1.7	122.0	....	....	....	....	....	....

a Charges dumped.

Main End Jackets.

Blast Furnace No. 2

Dec. 2, 1910.

Initial temperature of water 0.0° C. or 32° F. Each test was run on two east end jackets (upper and lower).

Test No.	Net Weight of Water. Pounds.	Discharging Temperature of Water. °C.	Elapsed Time. Min.-Sec	Avge.Temp. °C. As Read	Corrected. a	Rise in Temp. °F.	Pounds Water per Minute.	Gallons Water per Minute.	B.t u Removed per Minute.
				Observed Data		Calculated Data.			
I.	405.5	11.1 / 11.0	0– 0 0 / 0–44.1	11.05	10.75	19.35	551.7	66.15	10,675
II.	402.5	11.0 / 11.1	0– 0.0 / 0–44 1	11.05	10.75	19.35	547.6	65.66	10,596
III.	397.0	10.1 / 10.1	0– 0.0 / 0–43.4	10.10	9.80	17.6	548.8	65.80	9,659
IV.	395.5	10.0 / 10.0	0– 0.0 / 0–43–4	10 00	9.70	17.5	546.8	65.56	9,569
V.	408.0	10.1 / 10 1	0– 0.0 / 0–44.7	10.10	9.80	17.6	547.7	65.67	9,640
VI.	403.0	10.0 / 10.0	0– 0.0 / 0–43.9	10.00	9.70	17.5	550.8	66.04	9,639
VII.	411.0	16.1 / 16 0	0– 0.0 / 1–12.2	16.05	15.75	28.35	341.6	40.96	9,684
VIII.	411.5	16.0 / 16 0	0– 0.0 / 1–12.4	16.00	15.70	28.3	341.0	40.89	9,650
IX.	406.5	16.0 / 16.1	0– 0.0 / 1–11.1	16.05	15.75	28.35	343.0	41.13	9,724
X.	409.5	16.8 / 16.5	0– 0.0 / 1–11.9	16.65	16.35	29.4	341.7	40.97	10,046
XI.	407.0	17.0 / 16.9	0– 0.0 / 1–11.9	16.95	16.65	30.0	339.6	40.72	10,188
XII.	412.5	18.0 / 18.0	0– 0.0 / 1–12.6	18.00	17.70	31.9	340.9	40.88	10,875
XIII.	385.0	66.4 / 66.8 / 66.9 / 66.9 / 67.1 / 67.6	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0 0 / 4– 0.0 / 4–54.6	66.95	66.65	120 0	78.4	9.40	9,408
XIV.	377.5	68.7 / 68.8 / 69 0 / 69.0 / 69.1 / 69.0	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0.0 / 4– 0.0 / 4–51 2	68.93	68.63	123.5	77.8	9.33	9,608
XV.	384.5 / Furnace Charged.	70.0 / 70.0 / 69.9 / 69.5 / 69.8 / 68.5	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0.0 / 4– 0.0 / 4–55.6	69.62	69.32	124 8	78.0	9.35	9,734
XVI.	382.0	65.0 / 64.3 / 63.8 / 63.1 / 63.0 / 63.1	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0.0 / 4– 0.0 / 4–54.2	63.72	63.42	114.2	77.9	9.34	8,896
XVII.	393.5 / Furnace Charged	64.0 / 64.1 / 64.4 / 64.4 / 63.9 / 63–0	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0.0 / 4– 0.0 / 5– 2.1	63.97	63.67	114.6	78.2	9.38	8,962
XVIII.	396.5 / Furnace Charged	62 5 / 61.5 / 60.3 / 59.0 / 58.0 / 56.8	0– 0.0 / 1– 0.0 / 2– 0.0 / 3– 0.0 / 4– 0.0 / 5– 5.8	59.68	59.38	106.9	77.8	9.33	8,317

a Temperatures of discharge water taken with thermometer reading 0.3° C. too high. Column of Corrected Average Temperatures gives the true values.

Main Side Jackets.

Blast Furnace No. 2   Dec. 3 and 14, 1910.

Test No.	Date, December 1910.	Net Weight of Water Pounds.	Disch. Temp °C. Start of Test	Disch. Temp °C. End of Test.	Elapsed Time. Min -Sec.	Av'ge Temp. As Read.	Av'ge Temp. Corrected	Rise in Temperature °F.	Pounds Water per Minute.	Gallons Water per Minute	B.t u. Removed per Minute.
I.	3	408.5	4.2	4.1	0–43.5	4.15	3.85	6.9	563.4	67.55	3,887
II.	3	399.0	4.0	4.0	0–42.5	4.00	3.70	6.7	563.3	67.54	3,774
III.	3	401.5	4.0	4.0	0–42.5	4.00	3.70	6.7	566.8	67.96	3,798
IV.	3	413.5	4.0	4.0	0–44.0	4.00	3.70	6.7	563.9	67.61	3,778
V.	3	400.0	4.1	4.1	0–42.4	4.10	3.80	6.8	566.0	67.87	3,849
VI.	3	412.5	3.5	3.3	0–43.8	3.40	3.10	5.6	565.1	67.76	3,165
VII.	3	391.5	5.0	5.0	1–12.3	5.00	4.70	8.5	324.9	38.96	2,762
VIII.	3	418.5	5.0	5.1	1–16.7	5.05	4.75	8.6	327.4	39.26	2,816
IX.	3	406.5	5.1	5.0	1–15.0	5.05	4.75	8.6	325.2	38.99	2,797
X.	3	392.5	5.1	5.5	1–13.0	5.30	5.00	9.0	322.6	38.68	2,903
XI.	3	401.5	6.2	6.1	1–14.1	6.15	5.85	10.5	325.1	38.98	3,414
XII.	3	415.5	6.0	5.7	1–17.0	5.85	5.55	10.0	323.8	38.82	3,238
XIII.	14	385.5	....	....	3–6.4	58.7	59.0	106.2	124.1	14.88	13,179
XIV.	14	385.0	....	....	3–11.8	53.3	53.6	96.5	120.4	14.44	11,619
XV.	14	397.0	....	....	3–12.8	52.1	52.4	94.3	123.5	14.81	11,646
XVI.	14	383.0	....	....	3–5.8	56.5	56.8	102.2	123.7	14.83	12,642
XVII.	14	394.5	....	....	3–9.8	60.3	60.6	109.1	124.7	14.95	13,605
XVIII.	14	401.5	....	....	2–54.4	67.1	67.4	121.3	138.1	16.56	16,752
XIX.	14	374.0	....	....	3–1.6	33.2	33.5	60.3	123.6	14.82	7,453
XX.	14	387.0	....	....	3–10.4	26.5	26.8	48.2	122.0	14.63	5,880
XXI.	14	399.0	....	....	3–0.4	24.9	25.2	45.4	132.7	15.91	6,025
XXII.	14	382.0	....	....	3–11.6	22.4	22.7	40.9	119.6	14.34	4,892
XXIII.	14	377.5	....	....	3–6.4	22.1	22.4	40.3	121.5	14.57	4,896
XXIV.	14	373.5	....	....	3–6.0	22.8	23.1	41.6	120.5	14.45	5,013
XXV.	14	384.0	11.3	11.2	1–8.0	11.25	11.55	20.8	338.8	40.62	7,047
XXVI.	14	369.0	10.5	10.5	1–4.4	10.50	10.80	19.4	343.8	41.22	6,670
XXVII.	14	365.5	8.5	8.4	1–4.6	8.45	8.75	15.8	339.5	40.71	5,364
XXVIII.	14	369.5	8.6	8.6	1–4.4	8.60	8.90	16.0	344.3	41.28	5,509
XXIX.	14	369.5	9.3	9.6	1–5.2	9.45	9.75	17.6	340.0	40.77	5,984
XXX.	14	371.5	9.7	10.1	1–6.0	9.90	10.20	18.4	337.7	40.49	6,214

Detailed Temperature Readings, Tests 13 to 24, inclusive.

Initial temperature of water read 0.0 °C. Tests run on N. E. jackets.

Test No.	XIII.	XIV.	XV.	XVI.	XVII.	XVIII.	XIX.	XX.	XXI.	XXII.	XXIII.	XXIV.
Readings taken at	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.	Temperature °C.
Start of test..........	58.5	55.0	51.5	54.9	58.2	65.5	34.0	27.5	25.5	23.0	22.0	22.5
End of 1st minute..	58.8	52.5	51.7	55.8	59.7	67.0	33.1	26 5	25.1	22.5	22.0	22.6
End of 2nd minute	58.9	54.0	52.0	57.0	59.8	68.0	32.5	26.5	24.9	22.4	22.2	23.0
End of 3rd minute	58.5	52.3	58.0	57.5	62.0	68.0	33.0	26.0	24.0	22.0	22.2	23 0
End of test.	59.0	52.5	52.5	57 5	62.0	....	....	26.2	....	22.0	22.3	22.8

Main Side Jackets.

Blast Furnace No. 4.   Dec. 15, 1910.

Tests run on N. E. jackets. Initial temperature of water, 0° C. = 32° F.

Test No.	Net Weight of Water. Pounds.	Disch'g. Temp. Start—°C.	Disch'g. Temp. End of 30 Seconds.	Disch'g. Temp. End of Test—°C.	Elapsed Time. Min.-Sec.	Av'g. Temp. °C. As Read.	Av'g. Temp. °C. Corrected.	Rise in Temperature. °F.	Pounds Water per Minute.	Gallons Water per Minute.	B.t u. Removed per Minute.
XXXI.	410.0	.....	....	....	14– 1.4	37.6	37.9	68.2	29.24	3.51	1,994
XXXII.	381.5	.....	....	....	14– 0.4	38.8	39.1	70.4	27.24	3.27	1,918
XXXIII.	401.0	.....	....	....	14– 0.4	41.9	42.2	76.0	28.63	3.43	2,176
XXXIV.	386.5	.....	....	....	14– 1.6	51.3	51.6	92.9	27.55	3.30	2,559
XXXV.	368.0	.....	....	....	14– 0.2	52.9	53.2	95.8	26.28	3.15	2,518
XXXVI.	347.0	.....	....	....	14– 0.6	55.7	56.0	100.8	24.77	2.97	2,497
XXXVII.	387.0	9 9	9.9	9.9	1– 7.8	9.9	10.2	18.4	342.48	41.06	6,302
XXXVIII.	375.5	9.2	9.3	9.2	1– 5.8	9.23	9.53	17.2	342.40	41.06	5,889
XXXIX.	374.5	9.2	9.2	9.2	1– 5.6	9.2	9.5	17.1	342.53	41.07	5,857
XL.	370.0	9.0	9.0	9.0	1– 4.8	9.0	9.3	16.7	342.59	41.08	5,521
XLI.	368.0	9.0	9.0	9.0	1– 4.6	9.0	9.3	16.7	341.80	40.98	5,708
XLII.	372.0	8.6	8.5	8.4	1– 5.2	8.5	8.8	15.8	342.33	41.05	5,409
XLIII.	388.0	5.8	....	5.8	0–46.8	5.8	6.1	11.0	497.44	59.65	5,472
XLIV.	388.0	5.5	....	5.8	0–46.8	5.65	5.95	10.7	497.44	59.65	5,323
XLV.	387.0	5.8	....	5.8	0–46.4	5.8	6.1	11.0	500.43	60.00	5,505
XLVI.	384.0	5.5	....	5.5	0–46.0	5.5	5.8	10.4	500.87	60.06	5,209
XLVII.	392.5	5.5	....	5.5	0–47.0	5.5	5.8	10.4	501.06	60.08	5,211
XLVIII.	380.5	5.4	....	5.4	0–46.0	5.4	5.7	10.3	496.30	59.51	5,112

Detailed Temperature Readings, Tests 31 to 36, inclusive.

Test No.	XXXI.	XXXII.	XXXIII.	XXXIV.	XXXV.	XXXVI.
Readings taken at	Discharging Temperature. °C.	Discharging Temperature. °C.	Discharging Temperature. °C.	Discharging Temperature. °C.	Discharging Temperature. °C.	Discharging Temperature. °C.
Beginning of test....	36.9	39.6	37.7	47.8	55.5a	53.7
End of 1st minute,	37.0	40.2a	38.2	48.0	56.5	54.5
End of 2d minute,	37.3	40.5	38.7	48.5	55.0	54.9
End of 3d minute,	37.6	40.4	39.3	49.0	54.3	55.1
End of 4th minute,	37.9	40.3	40.0	49.5	53.5	55.2
End of 5th minute,	38.0a	39.9	40.3	50.2	52.7	55.0
End of 6th minute,	38.0	39.6	41.0	50.9	52.1	54.9
End of 7th minute,	37.9	39.0	41.4	51.1	51.8	54.9
End of 8th minute,	37.8	38.5	42.0	51.9	51.4	55.1
End of 9th minute,	37.5	38.0	42.6	52.6	51.4	55.6
End of 10th minute,	37.3	37.5	43.5	53.2	51.5	56.1
End of 11th minute,	37.5	37.1	44.5	53.5	51.6	57.0
End of 12th minute,	37.6	37.0	45.5	54.0	51.9	57.5
End of 13th minute,	37.8	37.0	46.5	54.6	52.1	58.0
End of test............	38.0	37.0	47.2	55.2	52.7	58.4

a Charges dumped.

Summary of Results.

Series (*i. e.*, a Group of Tests Run Under Similar Conditions)	Tests Nos.	Pounds Water per Minute. Average.	Gallons Water per Minute. Average.	Rise in Tem- perature °F. Average.	B.t u. Removed per Minute. Average.
		Nose.			
First series, hot discharging .	1–3 inclusive .	8.58	1.03	71.1	612
Sec'd series, warm discharging	4–6 inclusive .	23.50	2.82	39.6	930
Average	. . . .	. . .	. . .	. . .	771
		Spout.			
First series, warm discharging	1–3 inclusive .	71.67	8 59	61 9	4,439
Second series, hot discharging	4–6 inclusive	34.53	4.14	131.0	4,524
Third series, cold discharging	7–9 inclusive .	139.38	16.71	34.0	4,744
Average	. . . .	. . .	. . .	. . .	4,569
		Breast Jacket.			
First series, hot discharging .	1–3 inclusive .	9.80	1.17	85.3	834
Sec'd series, cold discharging.	4–6 inclusive .	67.59	8.10	15.8	1,066
Third series, warm discharging	7–9 inclusive .	33.10	3.97	42 5	1,407
Average	. . . .	. . .	. . .	. . .	1,102
		Main End Jackets.			
First series, cold discharging.	1– 6 inclusive.	548.9	65.81	18.2	9,963
Sec'd series, cool discharging.	7–12 inclusive.	341.3	40.93	29.4	10,028
Third series, hot discharging.	13–18 inclusive.	78.0	9.36	117.3	9,154
Average	. . . .	. . .	. .	. . .	9,715

Main Side Jackets.

The average B.t.u. removed per minute by the Main Side Jackets, as calculated from the entire eight series, amounts to 5,645. However, the individual series show such variations in this figure that the average is of little value, the furnace conditions having varied widely while the tests were being run.

First series, cold discharging.	1– 6 inclusive.	564.8	67.72	6.6	3,709
Sec'd series, cold discharging.	7–12 inclusive.	324.8	38.95	9.2	2,988
Third series, hot discharging.	13–18 inclusive.	125.8	15.08	104 9	13,241
Fo'th series, warm discharging.	19–24 inclusive.	123.3	14.79	46.1	5,693
Fifth series, cold discharging.	25–30 inclusive.	340.7	40.85	18.0	6,131
Sixth series, hot discharging	31–36 inclusive.	27.29	3.27	84.0	2,277
Sev'th series, cold discharging.	37–42 inclusive.	342.36	41.05	17.0	5,814
Eighth series, cold discharging	43–48 inclusive.	498.92	59.83	10.6	5,305

Average Heat Consumption of Side and End Jackets.

Taken from Tests of Nov. 3, and Dec. 2, 14, and 15, 1910.

Side Jackets.

Date of Test, 1910.	No. of Tests Represented.	Furnace No.	B.t.u. Removed per Minute by		
			Two Jackets.	All Jackets.	All Jackets. Approximate Figures for Computing.
November 3.................	4	4	26,099	104,397	....
December 14	18	2	8,355	33,420	....
December 15................	18	4	4,465	17,860	....
Totals and Averages...	40	.....	8,379	33,517	33,600

End Jackets.

November 3..................	7	4	16.120	32,240	....
December 2..................	18	2	9,715	19,430	....
Totals and Averages....	25	......	11,508	23,017	23,000

Effect of Length of Furnace on Heat Loss and Coke Consumption.

Based on average heat value of blast-furnace bin coke for December, 1910 ; 11,525 B.t.u. per pound.

Length of Furnace. Feet.	Hearth Area. Square Feet.	Total B t.u. Removed per Minute by			Equivalent Pounds Coke per Minute.	Pounds Coke for Equivalent Number of 15-Ft. Furnaces.	Ratio, Fuel for Combined Furnace to Fuel for 15-Ft. Furnaces.
		Side Jackets.	End Jackets.	All Jackets.			
							Per Cent.
15	70	33,600	23,000	56,600	4.91	4.91	100.0
30	140	67,200	23,000	90,200	7.83	9.82	79.7
45	210	100,800	23,000	123,800	10.74	14.73	72.9
51a	238	114,240	23,000	137,240	11.91	16.69	71.4
60	280	134,400	23,000	157,400	13.66	19.64	69.6
75	350	168,000	23,000	191,000	16.57	24.55	67.5
87a	406	194,880	23,000	217,880	18.90	28.48	66.4
90	420	201,600	23,000	224,600	19.49	29.46	66.2
105	490	235,200	23,000	258,200	22.40	34.37	65.2
120	560	268,800	23,000	291,800	25.32	39.28	64.5

a Sizes of Anaconda blast furnaces.

*

Percentage of Total Blast-Furnace Jacket Area Contained in End Jackets, for Various Lengths of Furnace.

Data.

Constant Width.—Between upper jackets, 6 ft. 0 in.

At tuyere level, 4 ft. 8 in.

Size of End Jackets.—Upper : Height, 7 ft. 5 in. ; width, 6 ft. 6 in.

Lower : Height, 7 ft. 5 in. ; width at top, 6 ft. 6 in. ; width at bottom, 5 ft. 0 in., overlapping side jackets, 3 in. on each side.

Size of Side Jackets.—Upper : Height, 7 ft. 5 in.

Lower : Vertical height, 7 ft. 5 in. ; slant height, 7 ft. 5.5 in. ; length and number according to length of furnace.

NOTE :—With a length between end jackets of 15 ft., the above dimensions are those of the present 56 by 180 in. furnace.

Length of Furnace.	Hearth Area.	Total Jacket or Radiating Surface.	Total Side Jacket Area.	Total End Jacket Area.	Square Feet of Radiating Surface per Square Foot Hearth.	Per Cent of Total Radiating Surface in End Jackets.
Feet.	Sq. Ft.	Square Feet.	Square Feet.	Square Feet		
15	70	613.00	446.12	166.88	8.76	27.2
30	140	1,059.12	892.24	166.88	7.57	15.8
45	210	1,505.24	1,338.36	166.88	7.17	11.1
60	280	1,951.36	1,784.48	166.88	6.97	8.6
75	350	2,397.48	2,230.60	166.88	6.85	7.0
90	420	2,843.60	2,676.72	166.88	6.77	5.9
105	490	3,289.72	3,122.84	166.88	6.71	5.1
120	560	3,735.84	3,568.96	166.88	6.67	4.5

The amount of radiating surface per square foot of hearth is reduced 20.4 per cent. by lengthening the furnace from 15 ft. to 60 ft., and 23.9 per cent. by lengthening it from 15 ft. to 120 ft.

The relative area of the end jackets is reduced 68.4 per cent. by lengthening the furnace from 15 ft. to 60 ft., and 83.5 per cent. by lengthening it from 15 ft. to 120 ft.

Development of the Basic-Lined Converter for Copper Mattes.

BY E. P. MATHEWSON, ANACONDA, MONT.

IN a discussion of a paper on The Basic Process as Applied to Copper Smelting, by Percy C. Gilchrist, read before the Society of Chemical Industry, London, Jan. 5, 1891,[1] Prof. W. C. Roberts-Austen asked Mr. Gilchrist whether he thought that the substitution of a basic lining for acid lining in the Manhès process would afford anything like the service which it had been shown to render in the metallurgy of iron.

Claude Vautin stated that he had experimented for over two years with basic linings for Bessemer converters for copper mattes at Cobar, but had given up the attempt on the score of cost. Mr. Gilchrist in his reply stated that he did not believe in applying any system of Bessemerizing to copper.

About the time of the presentation of the paper mentioned, Hermann Keller, Superintendent of the Parrot smelter, in Butte, was experimenting on a large scale with converters lined with magnesite brick. He gave up the idea on account of the cost of linings and because no particular advantage was observed. My belief is that his tuyeres and converters were too small.

A short time afterward similar experiments were tried at the Great Falls plant of the Boston & Montana Co., and at the Old Works of the Anaconda Copper Mining Co. in Anaconda. These were abandoned on the score of cost and the lack of advantages. The same cause of failure, in my opinion, holds here. The Anaconda Copper Mining Co., however, adopted the magnesite brick lining for its tilting casting machines.

Fig. 1 is a view of the first tilting furnace casting copper. It was made from an old Brueckner roasting cylinder. Fig. 2 shows an elevation and vertical and horizontal longitudinal sections of the second tilting furnace, built in 1897.

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[1] *Journal of the Society of Chemical Industry,* vol. x., No. 1, pp. 4 to 16 (Jan. 31, 1891).
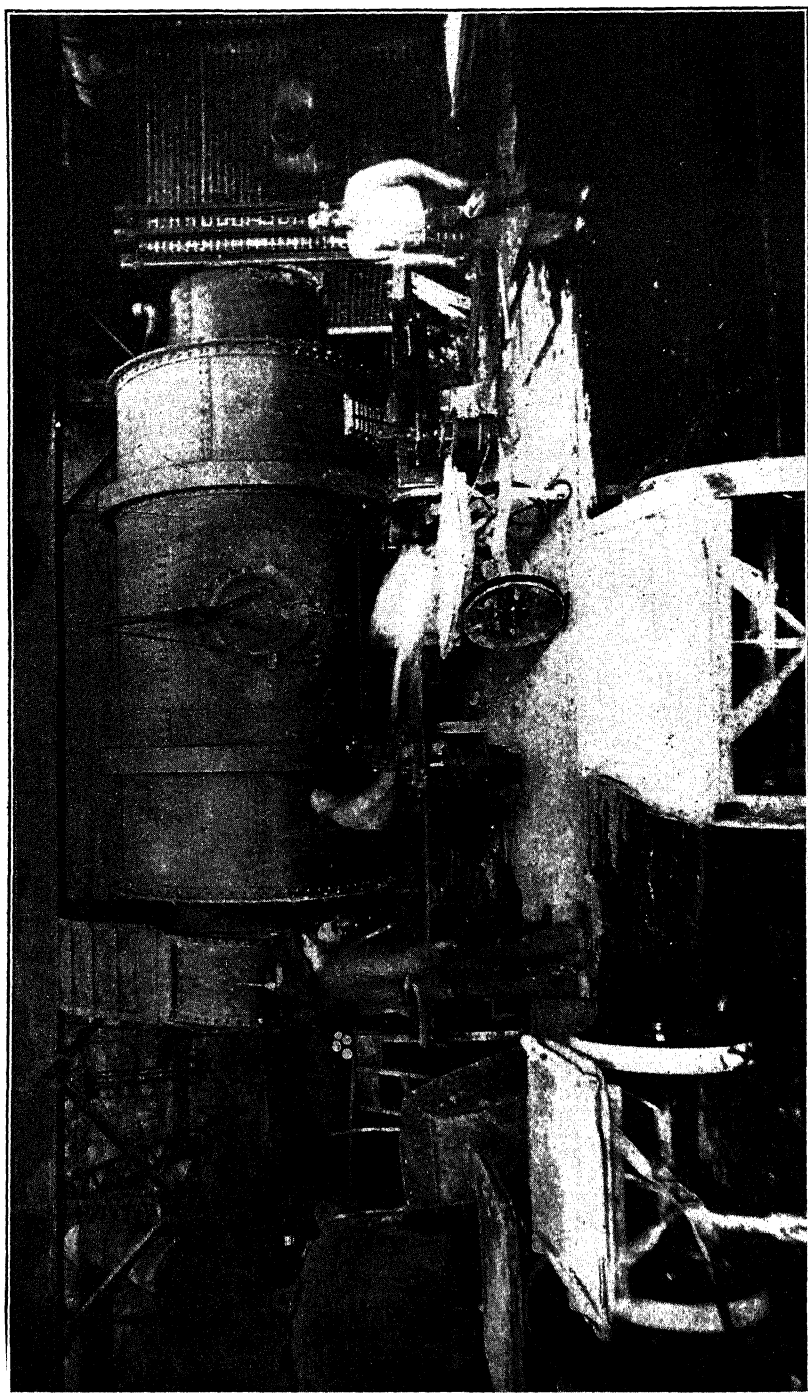
FIG. 1.—FIRST TILTING FURNACE OF THE ANACONDA COPPER MINING CO.
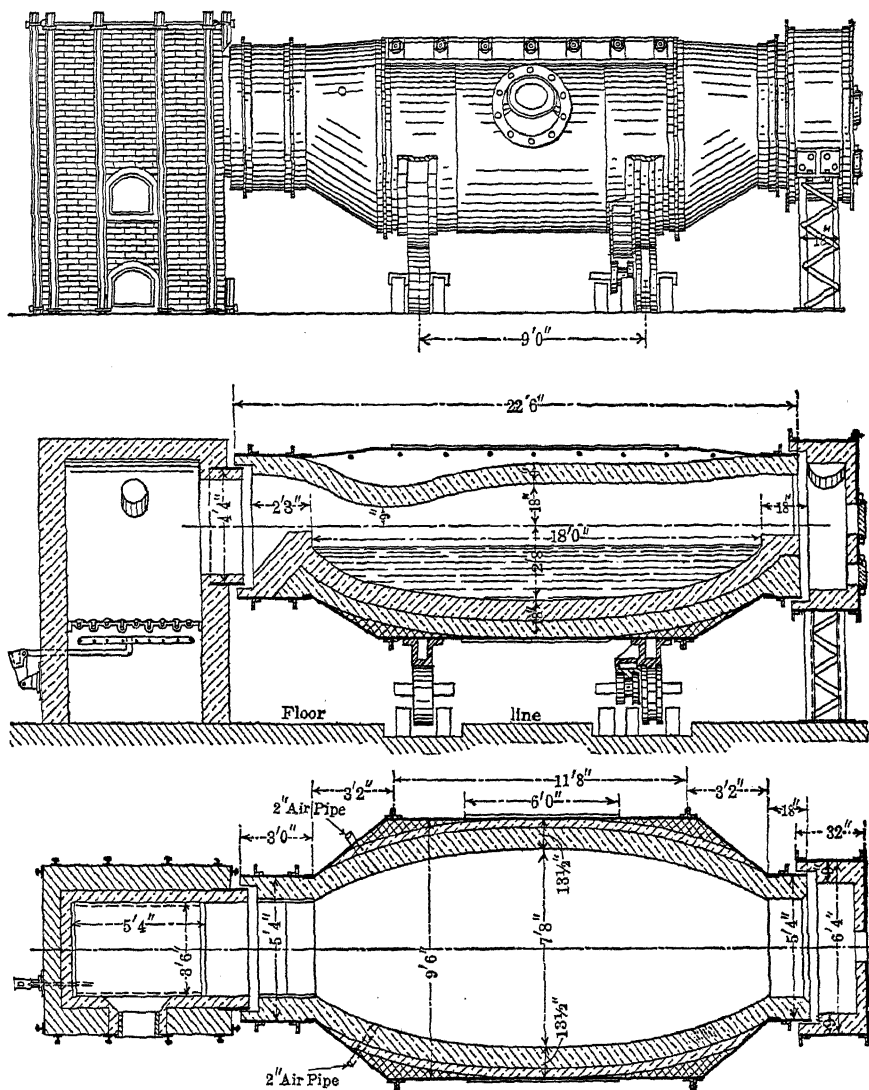
FIG. 2.—ELEVATION AND SECTIONS OF THE SECOND TILTING FURNACE OF THE ANACONDA COPPER MINING CO.

E. A. C. Smith, while temporarily in charge of these casting machines, tried the experiment of blowing the copper in the casting machines, but the return of the head of the department put a stop to the experiment, and Mr. Smith put the idea away till a more favorable opportunity presented itself.

About 1903 Baggaley began his experiments of smelting ore in a basic-lined converter in Butte, Mont., at the Pittsmont smelter. He was followed by Knudsen at the Sulitjelma plant, in Norway, in 1907. About the year 1905 similar experiments were tried at several smelters, notably the plant of the United States Smelting Co., at Midvale, Utah. Mr. Smith, who was then with the Baltimore Copper Co., found his opportunity to experiment, and his superintendent and manager, Mr. Peirce, gave him all the help in his power, the result being the construction at Baltimore of a basic-lined converter for leady copper mattes, along the lines of the old tilting anode furnaces of Anaconda. The experiment gave promising results, so the American Smelting & Refining Co. took up the process and introduced it with success on leady copper mattes at its lead refineries at Perth Amboy and Omaha. Then Messrs. Smith and Peirce persuaded the company to try it on straight copper mattes at the Garfield, Utah, plant.

In the meantime the Anaconda Copper Mining Co., at the Washoe Reduction Works, Anaconda, lined a shell of the standard trough pattern with magnesite brick. The results were excellent, so they gradually replaced all the acid lining with magnesite brick. At Great Falls the same company's experts developed a large converter along the lines of the upright shell, and, as this type is easier to build and keep in repair, it has become the standard during the past year. Practically all the Bessemerizing of copper mattes in the United States is now done in basic-lined converters. The main points to be observed for successful operation of the basic linings are: not to exceed a temperature of 2,100° F.; not to have tuyeres smaller than 1.25 in. (1.5 in. is the preferred size); to drive in punch rods the full size of the tuyere opening, immediately after pouring copper; to maintain in the converter as large a mass of matte and slag as possible to prevent sudden changes in temperature and overheating of the lining; to employ slag containing preferably about 25 per cent. of silica.

Magnesia in Basic-Lined Converter Slags.—The following test was made with a view to finding out whether the cutting action of converter slag on a magnesia brick lining bore any relation to the silica content of the slag.

Thirteen slags were selected from the daily samples sent to the

laboratory. These varied in silica from 22.4 per cent. to 37.8 per cent. and were carefully analyzed for MgO. The results show no relation between silica and magnesia content and are given in the following table:

TABLE I.—*Analyses of Slags.*

Cu.	SiO$_2$.	FeO.	CaO.	MgO.
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.2	37.8	46.3	1.1	0.6
2.2	37.2	46.6	1.0	0.6
3.80	35.1	48.3	1.0	0.6
2.80	34.5	49.3	1.0	0.6
3.20	33.5	50.3	1.0	0.5
3.80	31.6	51.4	1.4	0.5
1.80	30.8	52.5	1.3	0.8
2.20	28.9	56.0	1.0	0.5
2.40	28.0	56.5	0.9	0.5
3.00	25.8	56.6	1.0	0.5
4.60	24.6	56.2	1.0	0.7
4.60	22.4	58.3	1.3	0.7
4.20	23.4	58.1	1.1	0.4

To Smith and Peirce belongs the credit of taking a long-discarded idea and developing it into a successful process. The great advantages of the process are: decreased cost of lining; the ability to use large units in coverting, with consequent economies in labor, power, and repairs; neatness and cleanliness of plant, abolishing the danger, from dust, to the health of the lining crew.

DISCUSSION.

PROF. JOSEPH W. RICHARDS, South Bethlehem, Pa.:—Speaking to the question of the development of the basic lining, and speaking from a somewhat considerable study of the history of the process, I think Mr. Mathewson's conclusion (in the last paragraph of his paper) is perfectly justified and is very fairly stated. Messrs. Smith and Peirce worked at developing their process for many years; copper men had been trying their level best to get away from the silica lining of the converters and had uniformly failed. On one page of Mr. Mathewson's paper we read that at the old works of the Anaconda Company, at Anaconda, the experiments " were abandoned on the score of cost and lack of advantages." On the next page, after describing the work of Messrs. Smith and Peirce, we have the statement that the Anaconda Company lined a shell with magnesite brick and " the results were excellent." I think Mr. Mathewson's statement that to Smith and Peirce belongs the credit of developing a long-discredited idea, is correct.

C. D. Demond, Anaconda, Mont.:—It seems to me that the mortar used in lining is important. It is my impression that in basic steel furnaces tar is still used as a binder. At Anaconda, neither tar, linseed oil, nor molasses was at all satisfactory: but we have found that a solution of silicate of soda mixed with ground magnesite makes an entirely satisfactory and successful mortar with which to lay the magnesite bricks. Only a little of this mortar is used; just enough to make up for the inequalities in the bricks, which are laid close.

Hermann A. Keller, New York, N. Y. (communication to the Secretary *):—I have read Mr. Mathewson's paper on basic-lined converters with much interest. My experiments at the Parrot Works of Butte, which Mr. Mathewson refers to, I believe, were made in 1892, and were published by me in Dr. Peters's *Modern Copper Smelting.*[2]

The magnesite bricks then used were directly imported from Greece and naturally were very costly, and slow to obtain. I discontinued the practice at the time, because I considered our converter shells too small to hold sufficient amounts of heat without the aid of additional heating produced in slagging off the iron in the matte by the silica in the lining, thereby increasing the capacity of the converter with each additional charge until the lining had become so thin as to endanger the shell. The initial charges usually produced from three to five bars of blister copper, weighing about 110 lb. each, while it occurred frequently that the final charges of a well-made lining produced from 15 to 20 such bars. In order to increase our heat units still further we frequently resorted to the practice of throwing into the molten charge additional green matte and scraps, just prior to the final blow.

The size of the tuyeres may have considerable importance with basic-lined converters, as Mr. Mathewson points out, but I cannot speak from experience as to this point.

1891 and 1892 were years of considerable metallurgical improvement at Butte. Not alone did we modernize the converter practice during this period, but mechanical roasters of greatly increased capacity were also devised, besides constructing much larger reverberatory furnaces and installing coarse Harz jigs (known as bull jigs), with the double object of reducing concentration losses and to produce more material suitable for blast-furnace use.

Prior to the time of the above experiments the Parrot Works were about the only plant in this country that was converting copper upon

a large scale. This was done under the original Manhès patents, which consisted principally in raising the tuyeres some distance above the bottom of the shells, differing thereby in construction from those used in the metallurgy of iron, which latter were also much larger. The Anaconda Works at that time Bessemerized on a more limited scale and in different shaped converters, partly because they thought a more square shaped vessel would have greater advantage, and partly perhaps to circumvent, if possible, Manhès's patents under which the Parrot Company was operating. Cylindrical converters of much larger size were installed at Great Falls subsequently, and the barrel or Leghorn type of converters was introduced still later in this country. I might add here that some of the most recent installations have returned to cylindrical shells of the original Parrot type, only much larger.

From my experience I should consider that almost any kind of converter shell that is suitable for acid lining can readily be adapted to basic lining, provided it is sufficiently large to carry its own heat throughout the process. In fact, my belief is that the recent successes with basic converter linings are the direct outcome of the larger vessels, which are necessitated by the newer plants to handle larger quantities.

E. H. HAMILTON, West Norfolk, Va. (communication to the Secretary *):—In regard to Bessemerizing leady copper mattes: August Raht built a basic-lined converter at the Philadelphia Smelter, Pueblo, Colo., about 1895, and figured on catching fume in a long flue. I was present one day when the "pour" did not appear on the converter being turned over.

In 1900 Mr. Knox, at Ely, Vt., in attempting to operate a Garretson furnace (to make copper in the crucible of the blast furnace) lined the crucible with magnesite brick and predicted that magnesite would be the solution of the question.

A good deal has been written recently about detachable hoods to eliminate accretion, and also about flames. If a basic-lined converter plant is designed properly there will be no hood accretions, and if run to best advantage there will be practically no flame visible.

The quickest blown charges in acid-lined converters of which I have records might be said to be blown in the dark. Of course the fume, etc., from sulphur, lead, zinc, etc., is visible. The workmen object to this method as they want to see "something doing."

I would like to ask Mr. Mathewson what is the analysis of gases

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* Received Aug. 8, 1913.

from the basic-lined converter, as it affords an opportunity to obtain a long steady run under more uniform conditions than formerly in the acid-lined converters. Some years ago we found the oxygen content of the gas from matte in acid-lined converters to be as follows:

	Per Cent.
First half of the first blow	0.1
Second half of the first blow	1.0
First half of the second blow	9.0
Second half of the second blow	17.0

It would be interesting to know the percentage of nitrogen, SO_2, etc., from a basic-lined converter operated under present methods.

I understand from the paper that the MgO and CaO in the converter slag cannot be controlled, but that the Fe and SiO_2 can be controlled within certain limits. Can the gases be controlled? Varying the iron in the slag no doubt will affect the gas to some extent.

To what extent can converter gases from basic-lined converters be made available for the manufacture of sulphuric acid?

L. O. HOWARD, Globe, Ariz. (communication to the Secretary *) :— When we started running our 12-ft. converter we had a great deal to learn and the shell got some hard usage. The tendency at first was to use large charges which raised the back pressure and plugged the tuyeres. Later on, due to the men carelessly tilting the shell too far back when punching out tuyeres, copper ran into the pipes and we lost several more. Frequent renewals caused the brick to chip off in places and for a time it looked as if the wear at the tuyere line would be greatly in excess of any other part. This, however, did not prove to be the case and the accompanying diagram, Fig. 3, shows a good thickness of brick along this section when they had given out at other places. Doubtless the cooling action of the air helped.

In making this diagram the measurements were taken by turning the converter down and dropping a rod through the tuyeres until the end was just visible. When measurements *A* and *B* were taken, the covering seemed to have been well removed, but it will be noticed that considerable remained on at the end tuyeres. This, of course, made these figures worthless as a gauge for the wearing brick. Subsequently when measurements were taken, care was taken to remove all the coating and expose the naked brick.

Shortly after we had reached the 2,000,000 mark I stumbled across the fact that it was possible to coat the brick with magnetic iron oxide and thereby build up a protective coating. At one time we had, by actual measurement, 15 in. of this covering the brick, but it
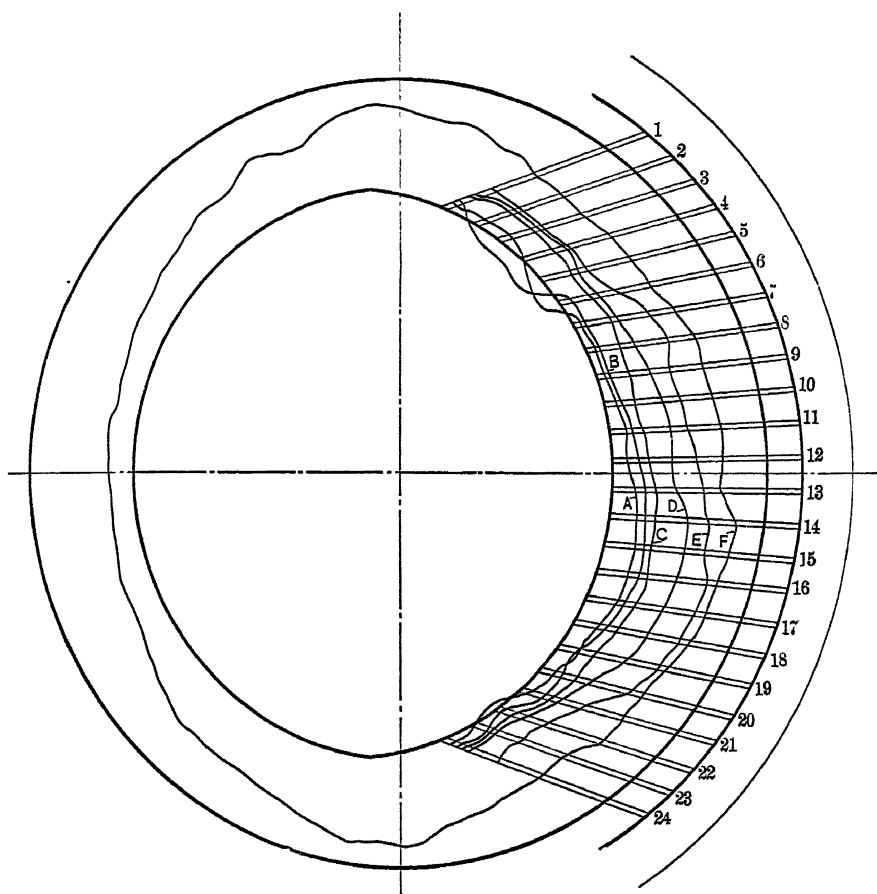
Measurements taken at intervals during life of lining to first patching.   Matte average = 43.9 per cent. of copper.

A.   After making   4,000,000 lb. of copper.
B.   After making   6,000,000 lb. of copper.
C.   After making   6,000,000 lb. of copper.
D.   After making 10,000,000 lb. of copper.
E.   After making 12,000,000 lb. of copper.
F.   After making 14,501,962 lb. of copper.

Brick was kept coated with Fe_3O_4.   Measurements A and B were taken with this coating in place.   For all subsequent measurements it was, as far as possible, removed. Brick finally gave out at back about 20 in. above tuyeres. Shell in operation from January 6 to June 29, 1913.

FIG. 3.—DIAGRAM SHOWING WEAR ON BRICK AT TUYERE LINE IN A 12-FT. GREAT FALLS TYPE BASIC CONVERTER.

so lessened the capacity of the shell that we had to take it off and put on a thinner one. Our great difficulty now is making the men keep the temperature down, as a short run with a high heat will clean everything off and expose the brick.

We have found it quite possible to keep this coating in place for many days at a time, during which period no sign of brick could be seen. Once with our first shell we ran for six weeks without a brick showing. Up to the present, however, we have been unable to keep a coating on indefinitely, as the men sometimes let a charge get very hot and melt it off. When too thick at the tuyeres it makes punching more difficult and for this reason it is not looked upon with favor by the help.

With 14,500,000 lb. of copper to the first patching, we should get 20,000,000 for the total life of the lining. Two places, one on each side of the converter, and a place 4 ft. long above the tuyeres showed greatest wear and gave out simultaneously. Later inspection showed that the lining had remained in place until less than 2 in. in thickness.

The shell was in continuous use for 173 days and made 664 charges, averaging slightly less than 11 tons of copper per charge. No transfers were made to it.

Our converting equipment consists of one 12-ft. stand (with which we endeavor to do all our work) and two 10-ft. 6-in. stands to be used as reserves. For the latter we have lined two of the old shells with magnesite brick. As our daily copper production is rather irregular, we sometimes find it necessary to relieve the 12-ft. shell by blowing one of the smaller size. This, however, is only done when the production runs over 50 tons per day for several consecutive days. The 12-ft. shell gets no rest between charges other than the time taken to punch out tuyeres and pull collars; perhaps one-half hour per charge. The record for this shell is 66.4 tons of copper in 24 hr. with a 52 per cent. copper matte, and 60 tons per day for three days with a 47.7 per cent. matte. For a period of three months we averaged slightly over 47 tons of copper per day. Matte averaged 43.8 per cent. Frequently we will run for two or three weeks without blowing the small shells. From this it will be seen that it is impossible to favor the converter either by transferring or by allowing it to cool between blows.

BANCROFT GORE, Butte, Mont. (communication to the Secretary *): —With reference to the small basic-lined converters installed at Gatico, Chile, February, 1912, I am pleased to present the following details:

These were of the barrel type; dimensions over all, 58 in. diameter by 84 in. long; hand-tilted by means of pinion and rack passing half round the converter at its center. The mouth was at the center, tapering to 28 in. in diameter.

For tuyeres, each was equipped with 13 hydraulic tubes, 1 in. internal diameter by $\frac{1}{4}$ in. thick, spaced 4 in. center to center and projecting 11 in. into converter. From the length over all we must deduct from each end $\frac{3}{8}$ in. for the thickness of the steel shell, $4\frac{1}{2}$ in. for one layer of magnesite arch brick cemented fast to shell by a thick mixture of sodium silicate and finely ground dead-burnt magnesite, and 6 in. for the protective coating of highly refractory basic slag formed under the conditions of operation and containing: CuO, 6.4; FeO, 78.5; SiO_2, 5.0; CaO, 1.8; Al_2O_3, 3.5 per cent. This gave a space about 5 ft. 4 in. long for the movement of the bath, or 8 in. between last tuyere and slag wall at each end. One layer of No. 1 arch brick was used throughout except for the region 12 in. above and below the tuyere zone, which had two layers of "straights." One shell was lined with two layers, making 9 in. of magnesite throughout, but it was found to have too small a capacity, especially after the growth of the protective coating mentioned above. This coating formed also on the bed of the converter and under adverse conditions would rise to within a few inches of the tuyeres. Blast pressure was 12 lb. No provision was made for the expansion of the brick, a matter of little importance in this size of converter.

The high conductivity of these brick cemented fast to the shell and the large area of this shell exposed to the cooling influence of the atmosphere, considering the relatively small bath, both favored the formation of this slag coating, which, when operating on a 45 per cent. copper matte, remained at a uniform thickness of about 6 in., thereby preserving indefinitely the life of the bricks except at the tuyere zone, where they were renewed about twice a year.

When the matte contained much less than 44 per cent. of copper the reaction temperature within the converter increased and the coating would gradually disappear, exposing the bricks, and at this stage their life was only a few days, unless the grade of matte could be raised to near 50 per cent., when a coating would rapidly form. After this protective coating was once in place the matte could be lowered to from 44 to 47 per cent.

Under normal conditions the converters took about 6,000 lb. of matte (in two charges), yielding more or less 2,500 lb. of copper at each pouring.

Without the protective influence of the slag coating and with the

bricks plainly exposed to view, the converter took as high as 22,000 lb. of matte (in three charges), yielding more or less 8,800 lb. of copper at a pouring. The speed of combustion working with this large bath of 40 per cent. matte increased very notably the temperature of the converter. Thus, when finishing for copper no punching was needed, as any noses forming were melted immediately. As mentioned above, the life of a lining under these conditions was only a few days.

Perhaps, if we had reduced the diameter of the tuyeres to $\frac{3}{4}$ or $\frac{13}{16}$ in., thereby slowing down the speed of combustion and restoring the balance between heat radiated and heat formed within the converter when blowing on these low mattes, the same slag coating mentioned above would have formed and saved the bricks. As our smelting conditions at Gatico seldom gave low mattes owing to the abundance of oxidized ores on the charge, we did not try to perfect a small converter that could handle these mattes.

As these magnesite bricks at Gatico cost us about 25 c. U. S. each, and served only as a backing for the growth of the basic coating, which after the first blow hid them permanently from view, it seemed possible that some substitute might be used, and in this connection I thought of the much cheaper " bauxite " brick. I would like to hear from some one who has tried them in furnace work, especially at points where both chrome and magnesia have failed to make good.

I would not advise the construction of converters smaller than those at Gatico, as even a few accidental charges of matte containing over 52 per cent. of copper would form such a thick coating of slag at the ends and on the bed that several tuyeres would be lost and the capacity reduced to the point where low-grade matte could not come to the rescue owing to the lack of sufficient mass reaction.

RALPH BAGGALEY, Pittsburg, Pa. (communication to the Secretary *) :—I am sorry that I cannot agree with Mr. Mathewson in his conclusions. I cannot, for the reason that I personally know that these conclusions are not correct.

The basic-lined converter and the feeding of silica (either mineral-bearing or barren) into the matte, in lieu of destroying the silica lining, are so closely linked that they may be treated as twins.

At the time I took up a study of the metallurgy of copper (about 1900), all of the authorities—Dr. Hermann Keller, Dr. Peters, Dr. Douglas, the Amalgamated experts, the Guggeheim metallurgists, etc.—pronounced the dissolving of ores in matte impossible. Mr.

Hixon called it suicide.   Dr. Keller has several pages of explanations in Dr. Peters's book as to just why it was impossible.

John Hollway tried it in 1878, and he met with considerable success.   If his backers (the Matthewsons of Wales and London) had provided him with such a converter as he then described (*i. e.*, with the tuyeres elevated on the sides and a quiescent pool below, in which the copper could accumulate without clogging his tuyeres), he would have succeeded beyond any doubt.

Four years later (1882), Pierre Manhès made a converter, as fully described by John Hollway in 1878, and forsooth called it "the Manhès converter," and obtained patents on it in France, England, the United States, etc.   In 1882, Manhès attempted to feed silica into the matte, both through the tuyeres and into the nose of the converter.   He scored a complete failure in both.   The former is the present J. Parke Channing process, with which Miami concentrates are now treated.

The sand-blast action is injurious to the tuyeres and the values are not extracted by the matte.   In order to recover these values, a subsequent melt of the slag in a blast furnace, or in some other form of furnace, is necessary.   Why then repeat Manhès's failure?

I invented the so-called Knudsen process before he did and I discarded it because of its defects and its useless expenses.   I fully described all of these things in an article that I wrote for the Anaconda *Standard*, Aug. 13, 1905.   On June 21, 1890, Jared E. Gaylord applied for a patent both on a mineral-bearing silica lining and on feeding ore into the matte; in the latter he failed completely.

Dr. Keller then attempted the same thing at the Parrot smelter, Butte, and failed.   His own account of this failure is given in Peters's *Modern Copper Smelting* (1895 ed.), p. 570 *et seq.*   The same thing was tried at Great Falls and failed.   The same thing was tried by Hixon at the old Marcus Daly smelter, Anaconda, and in his book (p. 81) he says: "The silicious lining is as much a part of the process of copper converting as magnesia lining is in the basic Bessemer treatment of phosphoric iron, and it is suicidal to attempt any other kind of lining, either water-jacketed or basic.   The improvement, if there is to be any, is to be in the line of mechanical devices, and the use of silicious ores to replace the expensive quartz and clay linings."

This must be the experiment for which Mr. Mathewson gives E. A. C. Smith credit.   However, it is not important just who conducted this experiment.   The result was a total failure and it simply confirmed the general belief that this new art was impossible.

This was the "state of the art" the world over at the time I com-

menced work at Butte. There were absolutely no basic-lined con-
verters in use for converting copper mattes and absolutely no one was
feeding and dissolving ores in a bath of matte. Many had tried it
and all had failed. The nearest approach to it was the use of a min-
eral-bearing lining, at Aguascalientes and as patented by Gaylord in
1890. A lot of misleading stuff has been published about my work
while at Butte and since. The facts are:

1. The apparatus furnished me by my associates was small and
bought solely for experiment; neither furnace nor converters were
intended to make copper on a commercial scale. After I proved that
I could dissolve ores in a basic-lined converter, I was then given my
" Jumbo converter," in which I worked up every pound of ore I had,
without regard to its grade or silica contents. But even then I had
no furnace that was much bigger than a bake-oven, or in which one
could afford to make copper commercially.

2. We bought up a lot of patents, such as Allen's, Potter's, Gaylord's,
Garretson's, etc., solely to get them out of the way of patents I in-
tended applying for. It was not intended by me to use any of them.
I well knew that they had all been tried and had failed. Garretson
was a very hard man to deal with, and in order to get his patents out
of the way I was compelled to agree to test his process to a finish.
We attempted this at Ore Knob, with Bellinger in charge, but we
were compelled to admit, as Garretson claimed, that it was not an
actual test. We then let Bellinger make another test at Vancouver
Island. The whole plant was so defective that they had neither
steam nor air. Garretson still demanded his " pound of flesh as
nominated in the bond!" We then decided that I should comply
with the terms of his contract by conducting these tests myself at
Butte. With this object in view, we built the little furnace in which
I later (as soon as I had any ore) made five series of tests of Garret-
son's process, and then and there proved that it was inoperative and
that it would freeze the furnace solid every time it was attempted.
In other words, the blowing of a converting blast into 85 per cent. of
slag (an oxide) had a greater chilling effect than the heating of 15
per cent. of matte (a sulphide) could overcome. The situation was,
of course, aggravated by his column of ores, etc., standing in the
matte.

Dr. Herreshoff and I had discussed these things and he later said
that my tests had simply developed and made clear five or six more
defects in the Garretson process than either of us had been able to
foresee.

3. My troubles at Butte were due to the fact that I had no ore to

treat, because the acid water and a lack of money delayed the development of the ore.

4. As soon as I got my Jumbo converter and a blowing engine that would work without breaking down every day, I commenced making excellent blister copper, and I continued this for $8\frac{1}{2}$ months, until every pound of ore in our bins or that I could buy had been worked up. I did this work too with one single basic lining (see Heywood's report in *Engineering and Mining Journal*, Mar. 24, 1906).

This was the first time in the whole history of the metallurgy of copper that mineral-bearing ores were successfully fed into the matte, as a means of removing the iron through the formation of silicate of iron slags in lieu of destroying the silica lining. It also included, of course, the successful use of a permanent, basic-lined converter. When our workmen announced in Butte that I was making copper by this new process no one would believe it. Fritz Heinze was the first to investigate and he was also the first to use my new process. Later Mr. Thayer brought Mr. Mathewson to visit me; Mr. Mathewson no doubt still remembers our discussion. Later Mr. Mathewson sent his assistant to our Pittsmont smelter, who spent parts of three days with us learning all of the details of this new art.

I left Butte about eight years ago and I have never been back there. Shortly after my return to Pittsburg I received a personal letter from Dr. Hermann Keller, in which he asked me to meet him at the Engineers' Club in New York on important business. I did this, and he took me to the offices of the American Smelting & Refining Co. I was introduced to Anton Eilers. He at once sent a messenger for August Raht and two other experts, whose names I did not learn. They may have been Messrs. Smith and Peirce, although I do not know this to be a fact. At all events, the office was cleared of all save these five experts and myself. We then commenced a discussion on methods of treating copper, which extended over two days. They all disputed the correctness of my theories. I told them I had been actually practicing this new art for $8\frac{1}{2}$ months and that I had only used one single basic lining in all of that time. They then asked me to build a plant at one of their smelters and demonstrate the correctness of my theories, on their agreement to adopt this new process at 16 of their smelters. My reply was that unfortunately my old patents were owned, for the districts where most of their smelters were located, by the Pittsburg & Montana Copper Co. They took down the name and address, and later they bought the company's rights under some of these patents.

Some time after this I again met August Raht; this time as the

consulting metallurgist of August Heckscher. He told me that none of the Guggenheim experts could believe that I could dissolve ores or silica in a matte bath in a basic-lined converter. However, on my positive statement that I had been practicing this new art for $8\frac{1}{2}$ months and by it had produced more than $500,000 worth of copper, within 30 days after my visit they commenced a series of tests of my theories in their Baltimore smelter, and I am told these tests were made by Messrs. Smith and Peirce, who were employed at the smelter in Baltimore. These tests proved that my theories were correct and the Guggenheims then brought out, on my old patents as a basis, what they facetiously called the "Smith and Peirce process." This process was quickly introduced at Baltimore, Perth Amboy, Omaha, Salt Lake, etc. I can prove by such men as Arthur S. Dwight, Dr. J. B. F. Herreshoff, Robert S. Towne, William A. Heywood, and many others, that I alone produced this revolutionary change in the metallurgy of copper, and I did it in direct opposition to the recorded opinions of all the authorities. However, public acknowledgment of the facts in this matter has been published, and by the very authorities who for years had pronounced this feat impossible.

In Dr. Peters's *Principles of Copper Smelting*, p. 465, he says: "As Mr. Baggaley is the originator of the method and Mr. Heywood is the technical manager, their views regarding the process might, naturally, be expected to be less impartial than those of an outside engineer; but it is evident that they are doing regularly what the profession, in general, has regarded as impracticable; namely, converting a very low grade matte in a vessel having a permanent basic lining, and supplying it with silica from outside sources."

In the light of all the foregoing facts, what basis of truth is there for the statement that "To Smith and Peirce belongs the credit of taking a long-discarded idea and developing it into a successful process?"

What have Smith and Peirce invented or developed? I practiced the art with perfect success for $8\frac{1}{2}$ months, using a single lining, years before they even commenced to test the correctness of my theories and which theories all of their own experts disputed. As a well-known authority has stated to me, all of Smith and Peirce's patents for "improvements" on my process are really "steps backward." Their design and construction are such that it is impossible to hold their lining or tuyeres in place. Their practice, I am told, compels them to reline every 45 days, never over 60 days. Their attempt to prolong the life of their lining by their patented addition of 15 per cent. of cold ore really means "reduced efficiency!"

The business end of our old company was a failure from the start and the result has been that I have never made a cent out of any of my inventions and my 13 years of study and effort. In the present paper, even the credit for this revolutionary change is given to others. When I left Butte, my contract with the old company terminated, and since then (for eight years) I have been studying this subject and developing improvements. I wish, in this connection, to record my opinion that there is really no good converter, built on scientific principles, in existence to-day. My old Jumbo at Butte is the best one, but it would not stand the service in my latest inventions.

This problem will not be finally solved until my continuous process is used. The heats must not be held down to 2,100°, as is claimed in this paper, thus reducing the efficiency, but, on the contrary, must be driven to the utmost limit. Apparatus must and can be provided that will stand this severe service and it will not be such as Mr. Mathewson pronounces "standard" at Great Falls, with an expensive lining $2\frac{1}{2}$ ft. thick. A 12-in. lining or even an 8-in. lining is ample, providing the design and construction are such that it can be held in place absolutely and providing ample provision is also made for radiating and dissipating these excessive heats from the outside of the vessel, instead of doing what all are now doing,—attempting to regulate these on the inside of the vessel.

I can show how all of these things can be safely and cheaply done. I can also show a mechanical tuyere puncher that will do far better work than men can; will do it almost without cost—compared with hand-work—and will do this efficiently year in and year out. As the modern converter is equipped with from 24 to 68 tuyeres, the punching becomes a laborious and expensive task.

My latest converter is cheap to build; it is absolutely rigid and secure in every respect. I am certain the lining will last for a year and I believe much longer. My old lining lasted $8\frac{1}{2}$ months. The process is continuous, and an absolute limit can be placed on the corrosion of the lining and to any degree desired without in any way interfering with successful converting up to the final stage of blister. It is difficult to do these things and do them in a perfectly safe way.

Great Falls Converter Practice.

BY ARCHER E. WHEELER AND MILO W. KREJCI, GREAT FALLS, MONT.

The Boston & Montana Reduction Works at Great Falls, Mont., was formerly the reduction works of the Boston & Montana Consolidated Copper & Silver Mining Co., and continued as the reduction plant for the independent company until June 1, 1910, when the consolidation of a large number of the operating companies of Butte was effected. It then became merged in the Anaconda Copper Mining Co. as the Boston & Montana Reduction Department of the Anaconda Copper Mining Co.

The history of copper converting at this plant, while not covering the entire period of the commercial use of the process in this country, does cover by far the larger part of it, and as the operation of and appliances for the process have undergone many changes during the time, it may, perhaps, be of some interest to put the record of these changes into permanent form.

The plan followed in the preparation of this paper has been to give first a brief chronological record of the principal events and changes and then, even at the risk of some repetitions, to give more detailed accounts of some of these changes under separate headings.

The headings for the complete paper are as follows:

1. General Chronological Review.
2. Mechanical Manipulation.
3. Casting of Finished Copper.
4. Tamping of Linings.
5. Early Acid Linings.
6. Acid Linings With the Use of Ore.
7. Introduction of Flux Through the Tuyeres.
8. Bottom Blast.
9. Disposal of Converter Slag.
10. Development of Different Classes of Converters After Class I.
11. Basic Lining.
12. Number and Size of Tuyeres.
13. Cast Copper Tuyeres.
14. Efficiency of Air.
15. Size of Converter Mouth.
16. Height of Converter.
17. Summary.

GENERAL CHRONOLOGICAL REVIEW.

Ground was broken for the general concentrating and smelting plant in the spring of 1891. The first departments to start operations were the concentrator in March, 1892, and the Brückner roasting and reverberatory smelting departments in April, 1892. The converter department

started operations in August, 1892. There were no blast furnaces until April, 1893. Therefore the first work of the converters was on reverberatory matte.

During the interval from April, 1892, to August, 1892, while the reverberatory furnaces were producing matte, but before the converters were

SECTION A-B-C

SECTION G-H
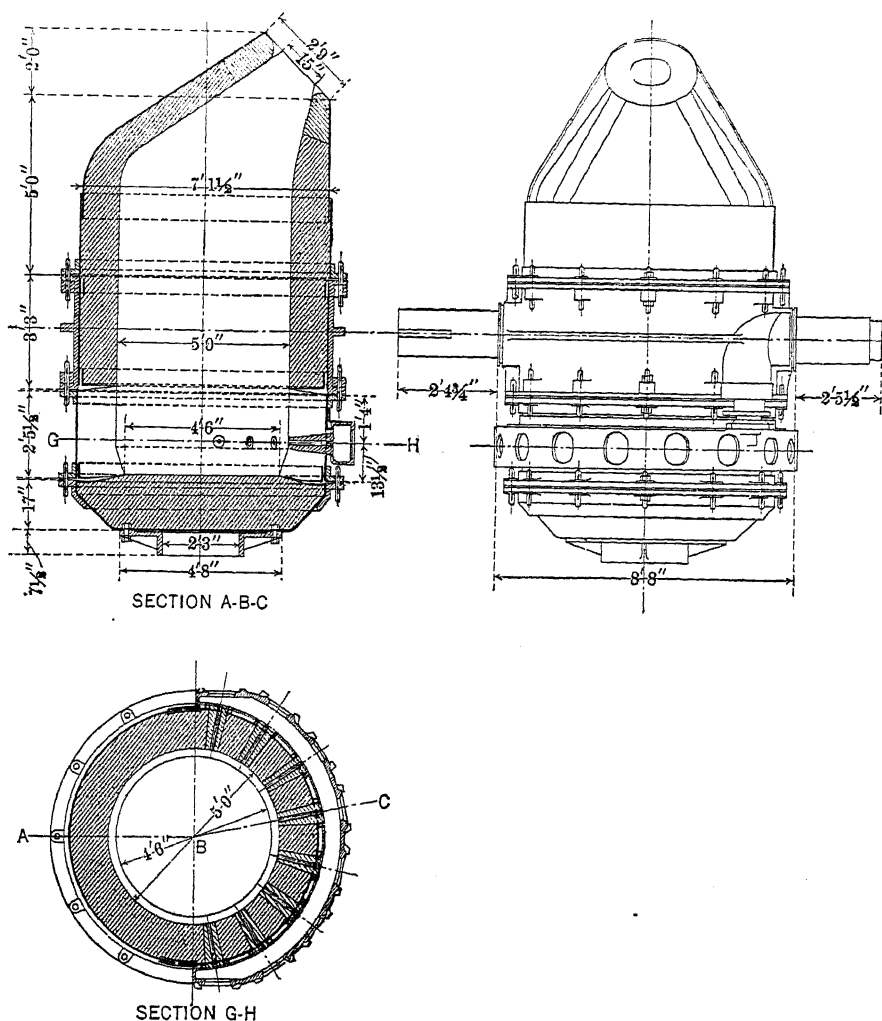
FIG. 1.—FIVE-TON CONVERTER.

installed, the matte was cast into molds, broken up when cold, sacked, and shipped to Lewisohn Bros., the predecessors of the United Metals Selling Co., at New York.

The first converters followed very closely in their design those used in steel practice, and were called five-ton converters. The design of this

converter is shown in Fig. 1. These converters were of the upright type, 14 ft. 9.5 in. high over all, circular in cross-section, 7 ft. 1.5 in. in diameter with eight side-blast tuyeres, and arranged for acid lining. They were tilted by hydraulic power, both the tilting and the manipulation of the air blast for the whole plant being controlled from a central "pulpit"; this method of control continued until 1899.

The plant started with two stalls. The converters were lined by hand, in place in the stalls, and it was soon found that the two stalls were not
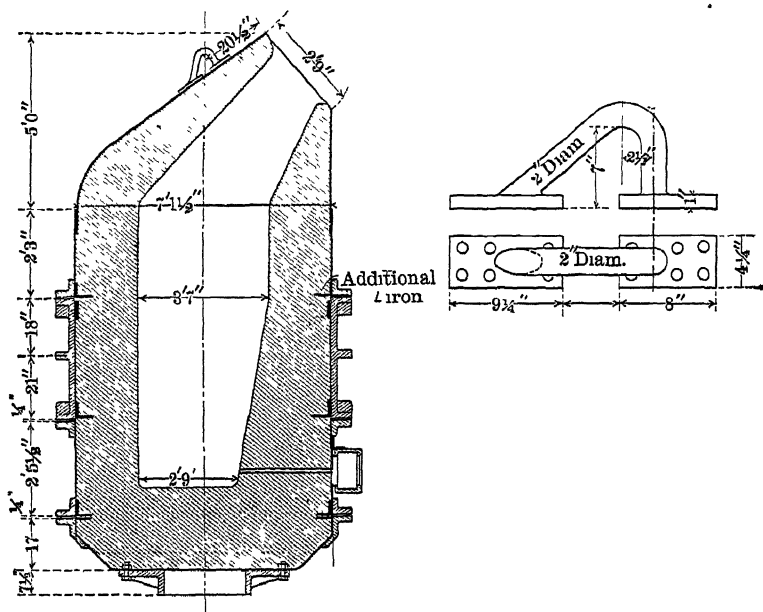
FIG. 2.—ADDITIONAL WORK ON CONVERTERS.

sufficient to allow proper time for drying and two more stalls were added of the same type as the first.

Whether or not the particular shape of lining shown in Fig. 1 was used is not recorded, but that it was not used long, if used at all, is evident from Fig. 2, which is from a drawing dated Mar. 18, 1893, seven months after the starting of the plant, which shows the shape of lining then adopted, and which in a general way, but with minor changes as to dimensions, was used as long as acid-lined converters were used. The chief characteristic is the thickening of the lining at the tuyere line. It was evidently early found that the greatest wear was at this point, and the lining was put in accordingly.

This same date, Mar. 18, 1893, is the first record of a plan for increasing the number of tuyeres, the drawing showing 17 instead of eight.

The first record of a design or attempt to put flux into the converter with the blast is a drawing dated Apr. 21, 1894, showing a sand box delivering sand into the wind box (see "Introduction of Flux Through the Tuyeres"). A trial of putting flux in with the blast was made in 1896.

In 1895 four more stalls were put in, making eight altogether, and all of the original type.

The earliest record which we find of a brick lining is a drawing dated Feb. 27, 1895, showing a Class I converter with a brick lining backed with ordinary lining clay. There is nothing to show whether the brick were acid or basic, nor is there any record as to whether or not it was ever tried. All we can say is that the matter had been thought of seriously enough to have a drawing made.

In 1894 plans were discussed for a machine for putting in the linings but it was not until about Apr. 1, 1896, that the machine had been put in and started.

In April, 1897, or perhaps in March, a basic (magnesite brick) lining was used, of which more detailed mention will be made under "Brick and Basic Linings".

In 1897 a converter was also operated into which ore was introduced through the mouth to furnish flux and save the lining

Another event of 1897 was the use of two bottom-blast converters which will be mentioned under "Bottom Blast".

On Dec. 4, 1899, a converter was lined with ore from the company mines and following this trials were made with linings two-thirds ore and one-third clay, and one-half ore and one-half clay. These linings were used until Apr. 20, 1900, when it was stopped until Aug. 27, 1900, after which linings of ore and clay were used until basic lining was adopted.

In 1900 the first change was made from central hydraulic and blast control for the plant, to individual control immediately at the converter.

In 1901 designs were begun for a converter slag casting machine, and it was completed and put into use on May 23, 1902.

In 1901 rights for the use of Walker copper-casting machines were purchased and on Mar. 31, 1902, a refining furnace for handling molten converter copper was started, this furnace delivering its copper to a Walker machine.

The furnace was abandoned on Jan. 6, 1903, and the machine used alone until a larger casting machine could be designed and built. This new machine went into service on Oct. 8, 1903.

In 1901 further trials of bottom blast were made.

In 1902 we began to discuss larger converters but it was not until Nov. 29, 1904, that the first Class II (larger than Class I) converter was actually put into service.

On Apr. 5, 1906, instructions were issued for the installation of the

second Class II stall, and extra bowls were built so that when this second stall went into service we had four of the Class II bowls or bottoms.

We felt that further improvement was possible and a larger converter called Class III was begun and the first one went into service on July 18, 1907.

On Jan. 21, 1907, a trial of smelting dried fine concentrates by dumping them into the converter was tried. Following this, in the same year, the blowing of these concentrates through the tuyeres was discussed and on Feb. 20, 1908, the drawings for an apparatus to do this were completed.

On Nov. 29, 1908, and following, this was tried, but never adopted as regular practice.

After this had been in operation some time we began the design of a larger converter called Class IV. The designs were completed on Mar. 24, 1909, but it was not until Feb. 4, 1910, that the first Class IV was put into service.

This became the standard and all Class I and Class II converters were abandoned. The Class III converters were retained as they gave good service and fitted into the same stalls as the Class IV.

On Mar. 9, 1911, the first run of what finally developed into the regular use of basic linings was started. It proved so successful that, as rapidly as possible, the department was changed over to basic lining.

Discussion was begun for plans for a still larger converter to be called Class V. This discussion resulted in the building of a converter 20 ft. in diameter, basic lined, and of the upright type. The first converter of this class was started on Aug. 3, 1912, and has proved entirely successful. While the Class IV converters are still in use the Class V is now considered the standard for the plant, and will be the only one used in the remodeled plant.

Experiments have been made on the use of different sizes of tuyeres, and on the efficiency of the air, and these subjects will be treated in a little more detail.

With this brief outline we will take up individually the subjects listed.

MECHANICAL MANIPULATION.

The original installation provided for the manipulation and tilting of the converters by hydraulic power. The converter revolved on trunnions, through one of which the air blast entered and was then carried by a passage cast in the trunnion section to the wind box. The other trunnion was fitted with a pinion which meshed with a rack attached to the piston rod of a hydraulic cylinder. The I-beams of the converter stand carrying the trunnion bearings formed the guide in which the rack moved back

and forth when driven by the piston, and thus rotated the converter in the usual manner.

The hydraulic pressure in the cylinders was normally 500 lb. per square inch. No pump was used at first, this pressure being obtained by supplying water under a static head of 125 lb. per square inch from tanks on a hill back of the smelter, to intensifiers which raised the pressure to 500 lb. per square inch.

The entire control of the air blast and of the tilting was from a central pulpit on the opposite side of the building from the converters and in full view of them. The converter crew directed the "tilter," as he was called, as to what was to be done by signals, much as train crews handle a train by signals.

Each converter was equipped with a four-way hydraulic valve, which, by the way, was of very complicated design, for rotating the converter in its stall, and also with a globe valve for the control of the air.

This system was used without change until the second set of two converters was installed, when the intensifiers were discarded and a steam-driven hydraulic pump was installed to pump directly into the main at 500 lb. pressure per square inch.

In order to provide ample water for the sudden demands of tilting several converters at once, an accumulator was installed which was simply a large piston of ample displacement, loaded on top with sufficient weight of cast iron, etc., to develop the pressure of the main over the area of the piston. This accumulator was connected to the main, and when the converters were not being tilted the pump would deliver into this accumulator, raising the piston to its full height before the pump would be stopped by the governor. Thus in case of a sudden demand for water at a rate greater than the capacity of the pump the accumulator would descend and force water into the main.

Mechanical difficulties were encountered and a great deal of work was required to keep this system in good working order. A complicated system of both water and air pipes was necessary, but except for improvements in details such as the substitution of a different type of water valve and many smaller details the general system was retained until 1900.

On July 25, 1900, a drawing was completed of an arrangement for the control of both the hydraulic tilting and the air blast immediately at the converter by one of the converter crew. With a few minor changes this proved to be very successful and convenient, and as rapidly as possible all converters were changed over.

This system continued in use without change until the installation of the first Class II converter. Electrical control of the tilting was installed on this converter and first put into use on Nov. 29, 1904, when the converter was first used, although the first conception of it was several months

previous. This is, we believe, the first application of electric control to a converter.

Very briefly, the converter was provided with a gear on one of its trunnions, this gear meshing with a pinion driven by a shaft on which was a worm gear meshing with a worm driven from the motor. Direct current at 500 volts was used.

No serious electrical troubles were encountered in the use of this tilting and as rapidly as new converters were installed, the electrical control was used, not only on the Class II converters but on all the larger classes following it.

CASTING OF FINISHED COPPER.

The original plan for casting the finished copper was to pour it from the converter into a ladle with a bottom discharge hole, this discharge hole to be plugged with a silica block on a rod which was protected by clay blocks strung on it as in the steel practice of casting ingots. The method was tried as planned when the plant started in August, 1892, but the plug very promptly froze in the hole and could not be pulled

After a few further attempts this method was abandoned and the method adopted of pouring the copper from the converter into a ladle, which was carried by the crane over molds set up on beams on the floor, and the copper tapped from a hole through the side of the ladle into the molds. This was not continued very long.

Until some other means could be provided the molds were set up on beams lengthwise of the converter floor; the copper was poured from the converter into a ladle hanging on the crane hook, and the crane then poured it into the molds which were turned over by hand; the copper was dumped onto the floor and taken away and trimmed by manual labor.

The next arrangement, immediately following this, was a spout curved vertically to fit the arc of travel of the mouth of the converter, this spout being mounted on wheels and designed to serve two converters standing side by side, by being set down by the crane in front of whichever one was to cast copper. The spout delivered the molten copper into molds which were mounted on a truck moved back and forth by hand on a track lengthwise of the converter floor. Each truck carried seven pig molds, or, when anodes were cast, four anode molds.

Considerable difficulty was encountered in keeping the track, which was entirely in the converter floor, free of obstruction, and the method also required considerable crane work. A change was made by using a somewhat similar spout to the one described above, but providing one for each converter stall, and hinging it on the stall, so that it could be swung into place by the converting crew when required. Drawings for this

change were completed and dated September, 1892. In Fig. 3 one of these spouts can be seen.

The trucks carrying the molds were run on tracks under the converter and at right angles to the converter floor. Pouring into the molds in both these cases was done directly from the converter by gradually tilting it.
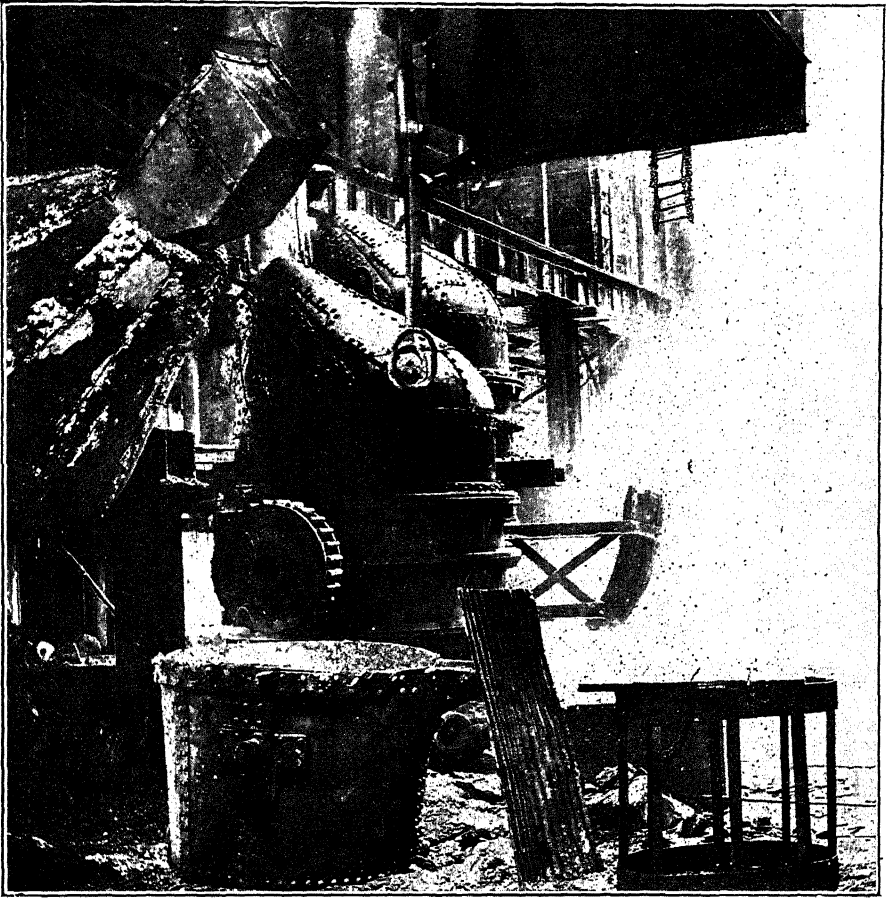
FIG. 3.—SHOWING CURVED SPOUT HINGED TO CONVERTER STALL.

When the molds were filled, the truck carrying them was run out on the floor of the "copper shed" back of the converters. The copper was allowed to cool, then dumped on the floor, trimmed by hand and trucked into railroad cars.

As the plant included an electrolytic refinery, the necessary anodes to supply it were cast, and the rest of the copper was cast as pig and shipped to Pawtucket, R. I., for refining.

This method of casting was continued until 1902. During this time it became evident that it was a crude, laborious, and uneconomical method of casting. It was hard and laborious, because the men were moving heavy trucks with heavy loads of hot copper over uneven floors on which obstructions of dirt, scraps of copper, etc., were constantly falling and blocking the wheels, and the work of removing the copper, particularly the anodes, from the molds was very heavy and hot work.

There was also a great deal of spilling and slopping of copper on the
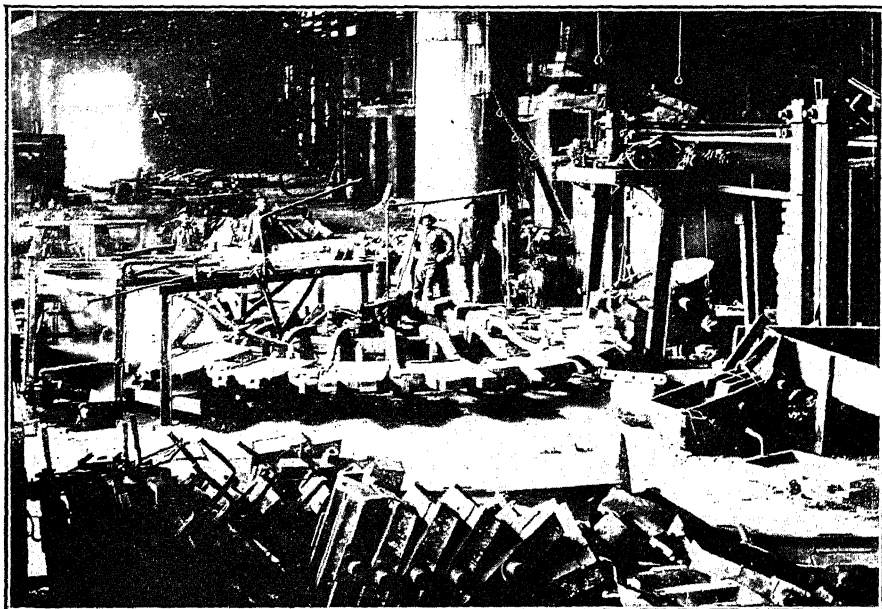
FIG. 4.—EARLY ARRANGEMENT OF CASTING MACHINE.

floor and over the edges of the molds, due to not being able to control the converter quickly enough.

A great disadvantage was the lost time of the converter. During all the time of pouring, which sometimes occupied as much as 1.5 hr. or more, the converter was not working. This amounted to a large percentage of the time and the elimination of this delay would add very materially to the possible production from a given equipment of converters.

Plans for ladles on trucks, pouring into stationary and also movable molds, were made as early as 1895 and 1896, but were not adopted.

In 1901 the trial was made of transferring the finished copper from several converters to one from which the casting was done. The idea was to lose the time of one converter only instead of each of them. It was found that the copper chilled in the converter to a great extent, due prob-

ably to its being exposed to the air while pouring from the first converter into the ladle, again in pouring from the ladle into the converter, and again the chilling of the small stream running from the mouth into a rather long spout for casting. Further than this the method did not improve the conditions of unloading the molds, and trimming and loading the copper.

In 1901 Mr. Klepetko, then manager, purchased the right to use some Walker casting machines. Plans were immediately begun for their use
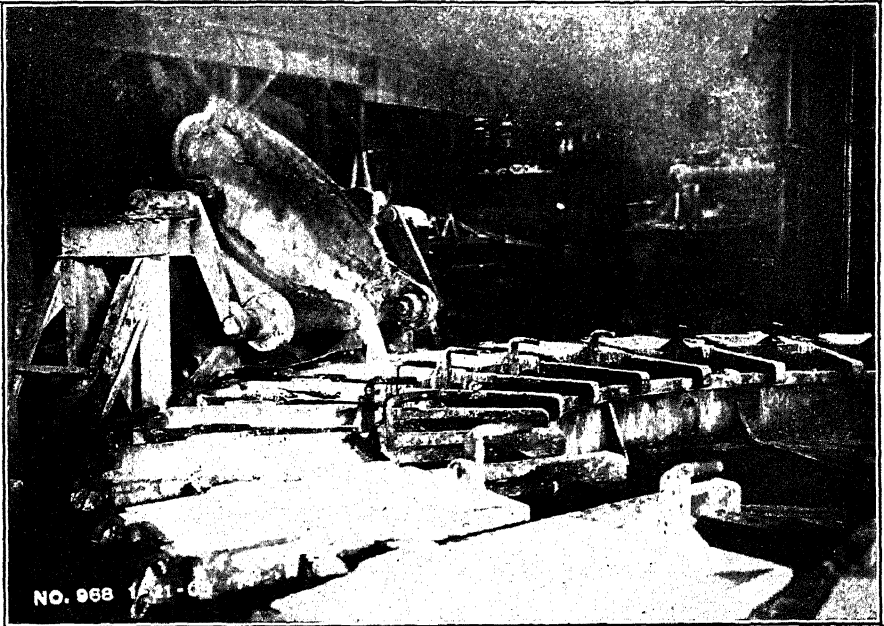
FIG. 5.—LADLE IN POURING POSITION ON CARRIAGE.

and the final plan adopted was to pour the copper from the converter into a ladle hanging on the crane hook. This was immediately carried by the crane to a coal-fired refining furnace and poured into it through a side door. As soon as sufficient copper had been accumulated, refining by rabbling and poling was begun.

As only one furnace was installed, it being the intention to await the results on this before erecting the second, the copper which was being produced by the converters during the refining of a charge was cast in the old way into pigs for shipment to the Eastern refinery. When the charge was refined it was tapped into the molds on the casting machine by allowing it to run over a bay which was cut down as required.

The first charge from this furnace was cast on Mar. 31, 1902. Considerable difficulty was encountered with both the furnace and the

machine. One of the principal difficulties with the machine was that it was not large enough to allow the copper to cool properly before dumping into the bosh.

Comprehensive experiments were being carried on at the electrolytic refinery at this time to determine whether or not there were sufficient advantages in the refined anode as compared with the rough converter anode to pay for the extra cost of refining. It began to develop that
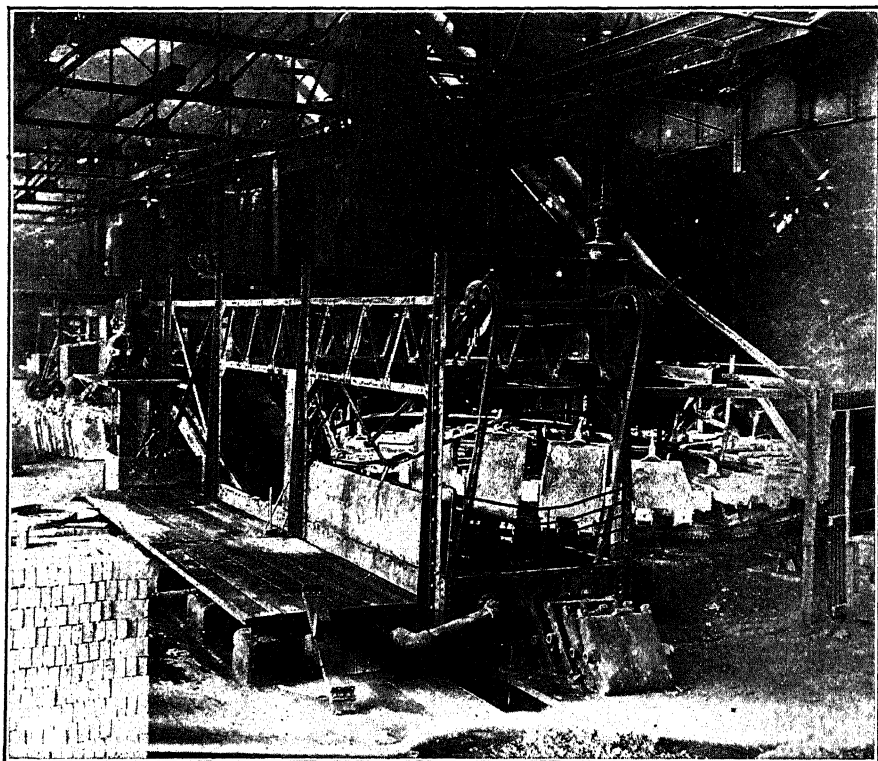
FIG. 6.—GENERAL VIEW OF CASTING MACHINE.

there were not sufficient advantages and therefore it was determined to abandon it. This was done on Jan. 6, 1903.

Arrangements were immediately made for the removal of the necessary parts of the furnace to allow a car carrying a ladle of molten converter copper to run from the converter floor to the casting machine. The copper was then cast by pouring it directly from the converter into a ladle which was placed on the transfer car by the crane, run to the casting machine and cast as rapidly as the size of the machine would allow. The first use of the machine in this way was on Mar. 30, 1903. A picture of the arrangement is shown in Fig. 4.

Designs were immediately begun of a machine of the Walker type, but very much enlarged and elaborated, which would allow the copper to be cast as fast as the stream could be poured from the ladle into the mold, and provide cooling time for this rate of casting. This meant primarily a machine of larger diameter. It was so placed that the crane could set a ladle of copper directly on the pouring carriage. This carriage was an entirely new arrangement hydraulically operated by a vertically moving piston, but the particularly novel feature was the
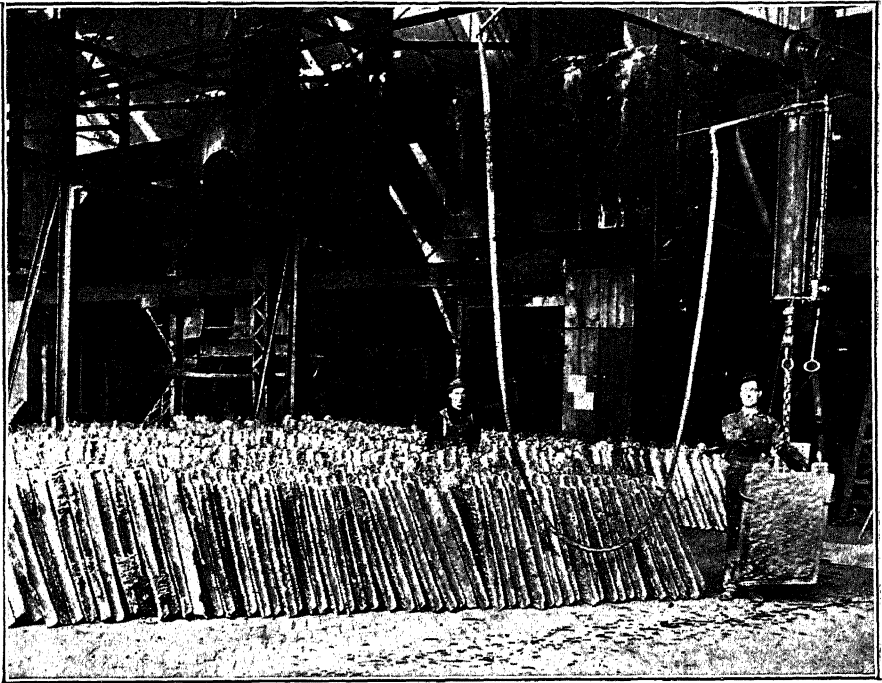
FIG. 7.—PNEUMATIC CYLINDER HOIST HANDLING ANODES.

double-hinging arrangement whereby the combination of all the motions produced the same effect as though the ladle were turning about trunnions located almost at the spout, thereby reducing the movement of the spout to such a small amount that the spatter and slop was very little. Fig. 5 shows a ladle in a pouring position on the carriage. Fig. 6 is a general view of the machine in action. The stream from the ladle can be seen at the right of the picture. The casting wheel is revolving clockwise and the molds with anodes which have just been cast and cooled can be seen in the center of the picture in different stages of dumping. On the left can be seen the anodes stacked on the floor. The copper is dumped automatically into a bosh containing water. It is carried up from this by

a drag conveyor and delivered to the copper-shed floor. Here it is picked up by means of a pneumatic-cylinder hoist which travels on the lower flange of an overhead I-beam 38 ft. long. One end of this beam is centered at the delivery end of the bosh and the other end travels through an arc of a little less than 90°, thus covering a large storage floor. Fig. 7 shows a view of this pneumatic cylinder in action and the anodes as they are stacked by it.

The machine is electrically operated, and is reversible in its motion, which was not true of the original machine. It was put into service on Oct. 8, 1903, and has been eminently satisfactory.

Three men per shift of eight hours, one manipulating the machine, one attending to the molds, etc., and one taking away copper, will easily cast, and stack on the storage floor, 200 tons or more of copper in 24 hr. On smaller quantities the machine is operated with less labor.

TAMPING OF LININGS.

The first linings were tamped by hand, with the converter standing in the stalls, the caps having first been removed. It early became evident that an improvement in the life of the lining could be made if more clay could be put in and also tamped in harder.

In 1894 plans were begun for a machine to do this work, and the machine was built, installed, and its use begun about Apr. 1, 1896. Fig. 8 shows a view of this machine with a converter bottom on it ready for tamping.

The cap was removed from the converter and the bottom was then taken out of the stall by the crane and set on a revolving table. Over this table was an air cylinder set in a frame allowing the cylinder to be moved by power up or down to any desired position and clamped. The frame and cylinder swung horizontally so as to cover the whole distance from the center of the converter to the circumference, so that the combination of all the motions covered the whole horizontal section of the converter, and the up-and-down movement of the cylinder allowed for the varying heights of the lining as it was being put in. The movement of the piston and rod which carried a tamping shoe on the bottom was controlled by the operator by means of a hand lever, an upward movement raising the piston, and a downward movement lowering it and striking a blow.

The machine was installed immediately in front of the clay-grinding machines so that the material delivered from them was shoveled directly into the converter bottom on the lining machine and tamped in. The lining was tamped solid, and the tuyeres punched afterward by driving a sharp-pointed bar through the lining for each hole. No attempt was

made to line the caps by machine. The machine was very successful
from the beginning and was used as long as acid linings were used.

A comparison of the linings by months in 1895 and 1896 is given in
Table I. Machine tamping began about Apr. 1, 1896.
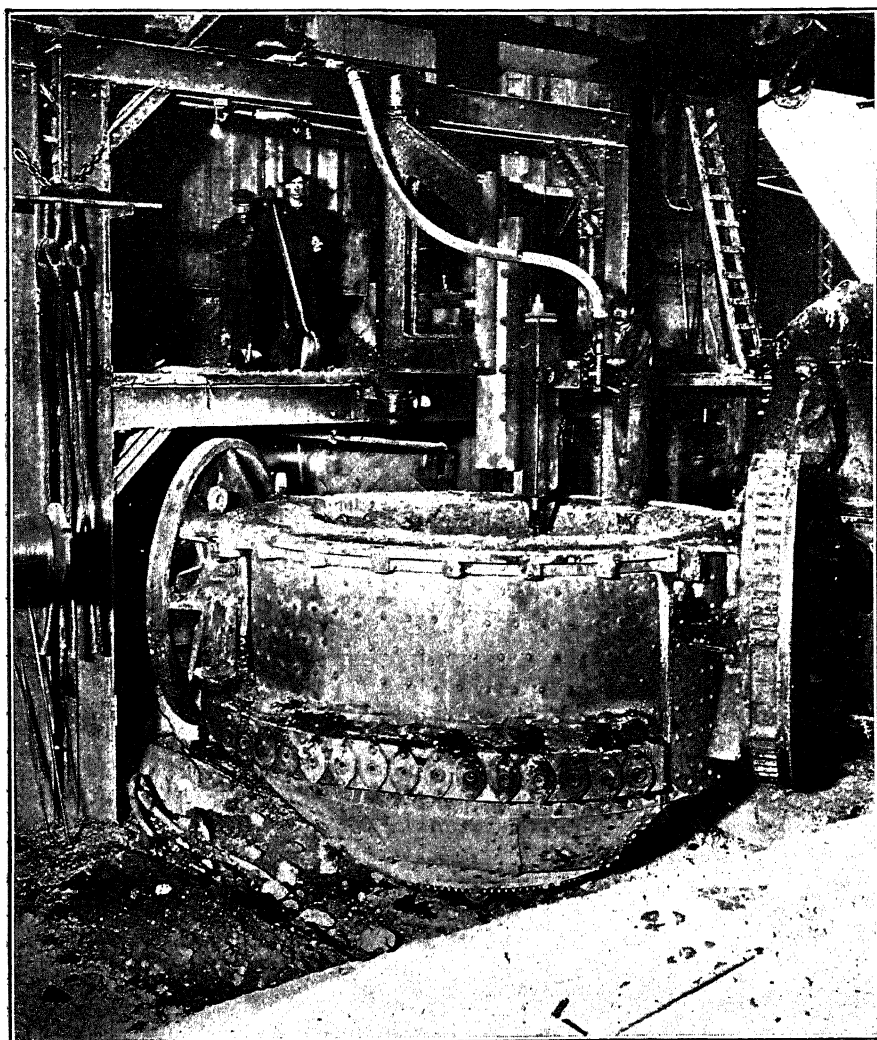
FIG. 8.—MACHINE FOR TAMPING LININGS.

Although the grade of the matte will average higher after Apr. 1, 1896,
than before, it is not, we think, sufficiently different to account for the
increased production per lining and the decreased consumption of lining
material per ton of copper. The apparent decrease in the tons of lining

material in one lining may be due to a change in the shape of the core around which the lining was tamped.

It appears that the decreased consumption of lining material per ton. of copper produced is due to the harder tamping of the material and the consequent decrease in caving in of the linings.

Just before the trials of basic lining were made, which led to the final adoption of them, a new tamping machine, striking a heavier blow, was

TABLE I.—*Comparison of Performance of Linings.*

Month	Grade of Matte Per Cent. Copper		Lb. Copper Produced Per Lining		Tons Lining Per Ton Copper Produced	
	1895	1896	1895	1896	1895	1896
January..........	52 5	50 3	30,416	26,840	0.385	0 505
February ...	48 9	51.6	26,046	26,740	0 485	0 536
March.	45.2	54.7	23,443	26,800	0.602	0.369
April	47 3	54.6	24,688	32,631	0.680	0.389
May	47 4	56 2	23,720	31,566	0.577	0.444
June	46.6	57 1	25,650	32,644	0.663	0 435
July	47.8	49.7	24,186	31,999	0.624	0 498
August	46.8	50 0	23,165	29,081	0.712	0.637
September	48.2	54.1	26,800	33,799	0 596	0.457
October..	48.4	57.2	28,902	38,561	0.453	0.407
November....	48.2	58.4	28,271	36,530	0.644	0 346
December...... .. .	47 9	57 4	25,848	34,101	0.526	0 512
Year.....	47.9	54.3	25,818	31,443	0.581	0 457

designed and had been partly built, but was never completed on account of the adoption of the basic lining as regular practice.

EARLY ACID LININGS.

When the first converter was started in August, 1892, the lining was a mixture of a siliceous rock from near Boulder, Mont., and a sand rock from near the bank of the Missouri river on the smelter property. Analyses of these materials are not available, but the Boulder material was a clean-looking pure quartz, and the sand rock was a more aluminous material furnishing the binding properties.

The lining was tamped into the converter by hand, with the converter in place in the stall. Whether or not it was tamped in the form shown in Fig. 1 we do not know, but that such shape was not continued long is certain, because in November, 1892, the center around which the lining was tamped was 3 ft. in diameter and set so as to make the lining much thicker on the tuyere side than on the opposite side.

During all the time that the original type or Class I was in use, the general shape of this lining was maintained with, however, changes from time to time in the thickness of the lining on the tuyere side on account of the faster wearing of the lining at this point.

During this early period a fair average performance of these converters was as follows, the figures being for 40 linings in January, 1895:

Total number of charges blown	217
Total copper cast, tons	582 488
Total time blowing on finish, min .. ·. 11420
Average number of charges per lining	5 425
Copper per average charge, tons.	2 685
Copper per lining, tons..	14 56
Time blowing on finish per charge, min.	53
Time blowing on finish per ton of copper, min.... .. .	19 6

The matte during this month of January averaged 53.2 per cent. Cu. The time quoted above is for the finish period only as this is comparable for all grades of matte and will be quoted in other parts of this paper for purposes of comparison.

The mixture of quartz and sand rock above referred to was used until October, 1895, when a sand rock obtained near the stack on the company property was substituted. This was known as B. & M. sand rock. An average analysis of this sand rock was: SiO_2 70.4; Al_2O_3 18.9; Fe 3.4; CaO 0.8; MgO 0.5 per cent. The use of this material was continued either for the entire lining or mixed with other materials as long as acid linings were used.

Acid Linings with Use of Ores.

The first use of ore in the converter was in 1895. At this time the rich pieces of copper glance were picked out at the mines, sacked, and shipped to the smelter separately from the other ores. This ore was thrown into the converter during the blow, but was not used in the lining.

In 1897 an extended experimental run was made of putting first-class ore into the converter through the mouth while the converter was blowing. The record of the experiment is given in Table II.

It is to be remembered that the converter in these runs was Class I side blast, acid lined, and the usual production of copper per lining was probably not over about 14.5 tons, from which it can be seen that the life of the lining was materially increased. The records as to why the practice was not continued are not complete.

On Dec. 4, 1899, the first converter, an original Class I, was lined with ore, unmixed with other material. Second-class West Colusa, that is the lower grade West Colusa, was used, the analysis of which was as fol-

lows: Cu 8.79; SiO_2 63.8; Fe 9.6 per cent. It was ground and handled the same as ordinary lining clay or sand rock, and tamped in with the machine in the same way as other linings. The converter was fired and dried in the usual way and put into service on Dec. 5, 1899. This ore

TABLE II.—*Experiment in Feeding Converter while Blowing.*

Date 1897	No. of Charges	Copper Produced Lb.	Ore Used Lb.
FIRST RUN			
Jan. 28		31,200	
Jan. 29		32,470	
Jan. 30		4,200	
Feb. 1		3,400	
Feb 2		8,400	
Feb 3		10,400	
Total for the run	. . .	90,070	
SECOND RUN			
Feb. 16 .	4	24,560	
Feb. 17	6	63,650	Not
Feb. 18	5	47,700	known
Feb. 19		30,580	
Total for the run	. . .	166,490	
THIRD RUN			
Mar. 3	3	12,140	
FOURTH RUN			
Mar. 15	3	15,150	
Mar. 16	1	3,400	
Mar. 17	5	21,200	8,100
Mar. 18	2	14,000	6,000
Mar. 19	3	14,600	18,790
Mar. 20	4	22,800	30,570
Mar. 21.	3	17,000	24,940
Total	21	108,150	88,400

lining was continued until Dec. 9, 1899, and a great deal of trouble was reported in making the linings last.

On Dec. 10, 1899, a change was made to a mixture of two-thirds second-class ore and one-third B.&.M sand rock. An analysis of this mixture was: Cu 7.0; SiO_2 60.4; $Al_2 O_3$ 14.66; Fe 6.3; S 9.6; CaO 0.3; MgO 0.5 per cent.

The linings continued to wear out more rapidly than those entirely of B. & M. sand rock and on Dec. 14, 1899, the mixture was changed to one-

half second-class ore and one-half B. & M. sand rock, and was continued so until Apr. 20, 1900, when the ore was discontinued for a time.

On Aug. 27, 1900, a mixture of second-class ore and B. & M. sand rock was again used, and from that time on, as long as acid linings were used, mixtures of sand rock with ore, refinery slag, electrolytic-plant refuse, copper precipitates from the mine waters at Butte, etc., were used in the linings. As much copper-bearing material as possible was used.

The practice of dumping ore as it came from the mine into the converter became the regular practice in February, 1905, and has always continued, both through acid- and basic-lining practice.

On Jan. 21, 1907, a trial of smelting dried fine concentrates by dumping them into the mouth on top of the charge when the converter was turned down was tried. Only two charges were tried. In the first one 1.5 tons of concentrates were used in a charge finally producing 6.05 tons of copper, and in the second one 2.5 tons of concentrates were used in a charge producing 6.79 tons of copper. There was, of course, some blowing out of concentrates when the converter was turned up, but no attempt was made to determine what this amounted to. The smelting was entirely successful and rapid.

In discussing the results of this test it was decided to try introducing the concentrates through the tuyeres and this will be mentioned under the heading "Introduction of Flux Through the Tuyeres."

INTRODUCTION OF FLUX THROUGH THE TUYERES.

Very early in the use of the converters it appealed to the management that the weak and expensive part of the process was the lining, and many suggestions and plans were made to improve this feature. One of the first suggestions was the introduction of siliceous flux through the tuyeres by means of the air blast.

A drawing was completed on Apr. 21, 1894, showing a sand box attached to the converter and discharging sand into the wind box from which it would be carried through the tuyeres and into the converter by the blast. This is shown as Fig. 9, and is the earliest record of any plan for this work.

We do not know whether or not this arrangement was ever put into service, but it is the recollection of men still connected with the plant that sand was blown in through the tuyeres as early as 1896. It was not continued more than a few days, as a great deal of trouble was experienced from the material blowing back into the punchers' eyes when the tuyeres were opened for punching.

No further attempts were made to introduce material through the tuyeres until after the experiment of smelting fine concentrates by dump-

ing them into the converter as mentioned under "Acid Linings with Use of Ores" was tried.

After the above experiment no actual steps were taken to continue the experimenting until the latter part of 1907, when plans were discussed and a drawing completed Feb. 20, 1908, of an apparatus for the introduction of these fine concentrates through the tuyeres. The proposed appa-

FIG. 9.—CONVERTER EQUIPPED WITH SAND BOX.

ratus was installed but it was not until Nov. 29, 1908, that it was tried, these trials being on regular acid-lined Class I converters.

It had been found by previous experiments that, on account of the tuyere opening being some distance above the bottom of the wind box, concentrates would accumulate in the box below the tuyeres, and when the converter was turned down these concentrates would run down into

each end of the wind box and plug the end tuyeres. To overcome this difficulty the bottom of the wind boxes of the two converters used in the experiments was filled with cement up to the tuyere openings.

During November and December, 1908, and January, 1909, 23 charges were run with the introduction of dried fine concentrates through the tuyeres. In the following tabulation only 22 charges are given, as no observation of the time of blowing was made for one of the charges.

As a matter of comparison 30 regular charges without concentrates are given for practically the same dates. The data for these latter charges were not especially obtained by observers as were those on the concentrate charges, but were taken from the regular daily reports.

The following is the analysis of the concentrates charged on Dec. 20, 1908, and is a fair analysis of those used during the entire run: Cu 8.45; SiO_2 20.2; Al_2O_3 4.7; Fe 28.2; S 34; CaO 0.2 per cent.

The tests were run more as qualitative tests to determine whether or not the scheme would work than as quantitative tests to determine the relations between matte, concentrates, etc., but nevertheless the results are quoted below:

	Regular Charges	Charges With Concentrates
Number of charges..........................	30	22
Total copper produced, tons	84 7	49.45
Total time matte to white metal, min........ ..	3,162	2,762
Total concentrates used, tons.	None	47.594
Copper per charge.....	2.823	2 248
Minutes blowing per ton of copper....	....	....
Matte to white metal........	37 3	55 9
Concentrates used per ton copper produced.......	None	0.9614

A thermal calculation on matte and concentrates made previous to these experiments showed a deficiency in heat which would be produced by the oxidation and chemical reaction of the converter charge of matte and concentrates when the concentrates were used in the proper quantity to produce the right slag. Accordingly in these experiments quantities of coke varying from 1 to 2.4 per cent. of the weight of the concentrates were used on the different charges.

From the above it will be seen that when concentrates are used the elapsed time of blowing from charge to white metal is greater per ton of copper than when concentrates are not used. This is on the assumption that the grade of the matte is the same in both cases and also that the lost time is the same. The grade of the matte and the lost time for the concentrate charges were observed and recorded, but as no corresponding data for the regular charges were available, it is not necessary to tabulate this information here.

With the assumptions made above we may calculate the extra time consumed in smelting the concentrates as follows:

Copper produced from charges smelting concentrates, tons	49 45
Time blowing per ton of copper in regular charges, min. ..	37 3
Estimated time for 49.45 tons, min.....	1,844
Time blowing concentrate charges, min..	2,762
Excess time for blowing concentrates, min.	918
Excess time for blowing per ton of concentrates, min....	19.3

A fair average of the slag skimmed from the charges in which concentrates were smelted is: SiO_2 23.7; Al_2O_3 6.1; FeO 60.3; CaO 0.7 per cent. This is practically the same as the slag produced by a converter running on a regular charge.

The feeding of the concentrates was from a closed tank discharging into the air main. Several mechanical difficulties were encountered which pointed the way to a better mechanical arrangement had the experiments developed into regular practice.

It was observed that the concentrates produced a marked chilling effect on the charge, particularly when the grade of the matte was high. The addition of the small percentages of coke mentioned produced some effect, but the greatest factor in heat production was the grade of the matte. This chilling of the charge produced attendant troubles such as heavy working, fouling of the mouth, etc.

Another difficulty was the mechanical wear of the tuyere holes in the lining. As the tuyeres in one of the bottoms used were badly spaced, that is, two of them opened into the converter near together, the enlarging of the original holes soon brought these two holes together and resulted in a great deal of throwing out of material at the mouth, with consequent fouling of the mouth, and delays to clean away the accumulation. This lost time is included in the time per ton of copper for blowing and of course tends to make it abnormally high.

It was also found that the concentrates cut through the steel tuyere pipes carrying the air from the wind box into the converter, and even cut down into the shell of the converter. A new converter, which was being built at the time these experiments were going on, was made with large tuyere pipes and removable cast bushings to take care of this wear should the process be adopted.

The rate of feeding concentrates varied from 52 to 210 lb. per minute, based on the time of actual feeding, which was intermittent. The average rate based on the total elapsed time from charge to white metal was 34.4 lb., or 0.0172 tons. The total elapsed time of blowing from white metal to copper was 1,199 min. With 2,762 min. from charge to white metal, the elapsed time from charge to copper was 3,961 min. Based on this

time the rate of feeding concentrates was 24 lb., or 0.012 tons per minute.

It should be borne in mind that this was an experimental run with only a small amount of previous trial of the manipulation of the process, and was carried out in a small converter of a class which has since been abandoned. With the present basic-lined converters of a much larger size, there is no doubt but that much better results can be obtained, particularly with practice.

The further trial of this process was dropped, however, as other processes were under consideration for trial, and development.

BOTTOM BLAST.

The first drawings for the converter plant show both side-blast and bottom-blast converters, but the first installation was side blast. No trial of bottom blast was made in a commercial way until 1897, or 1898.

On July 17, 1897, a drawing was completed of a bottom-blast bottom section to fit the upper part of a regular Class I converter. The tuyeres were of course vertical. Two bottoms were made, one with a single row of 17⅜-in tuyeres spaced on the center line passing through the trunnions, and another with three parallel rows of the same size tuyeres spaced on this center line and a line on each side of and a few inches from it.

The converter with three rows of tuyeres was used several times blowing complete charges, demonstrating that bottom blast could be used commercially. The converter was not punched while blowing, but while it was turned down and with no metal over the tuyeres. The wear of the side lining was observed to be very uniform, much more so than on a side-blast converter. The operation of this converter did not appear at all impossible, but was not as convenient as the side blast and was abandoned.

In February, 1901, another attempt at operating bottom blast was made, which as it developed might be considered somewhat in the light of a trial of basic lining.

A regular Class I converter was fitted with a 9-in. magnesite-brick bottom close to the shell. On top of this 18 in. of regular clay was tamped and then the regular clay (acid) lining continued up the sides. The converter, having been previously dried and heated, was put into service on Feb. 3, 1901, being charged with matte carrying 55.1 per cent. of copper. The clay bottom floated up on the first charge, and this charge and the balance of this campaign were finished on the magnesite-brick bottom. The converter made 22 tons of copper before the side lining was worn out as against 15.9 tons of copper for other regular acid-lined side-blast converters during the same period.

The converter was then relined with clay as on the first campaign and put into service on Feb. 6, 1901. The clay bottom again came up on the first charge and the campaign was finished on the magnesite-brick bot-

tom. On this campaign the converter made 18.8 tons of copper from a matte carrying 48.5 per cent. of copper, as against 14.4 tons of copper per lining from a 48.1 per cent. matte by the other acid-lined side-blast converters.

This is the extent of the trials of bottom blast at this plant.

Disposal of Converter Slag.

For a considerable time after the starting of the plant the converter slag was poured, while molten, into the reverberatory furnaces. When
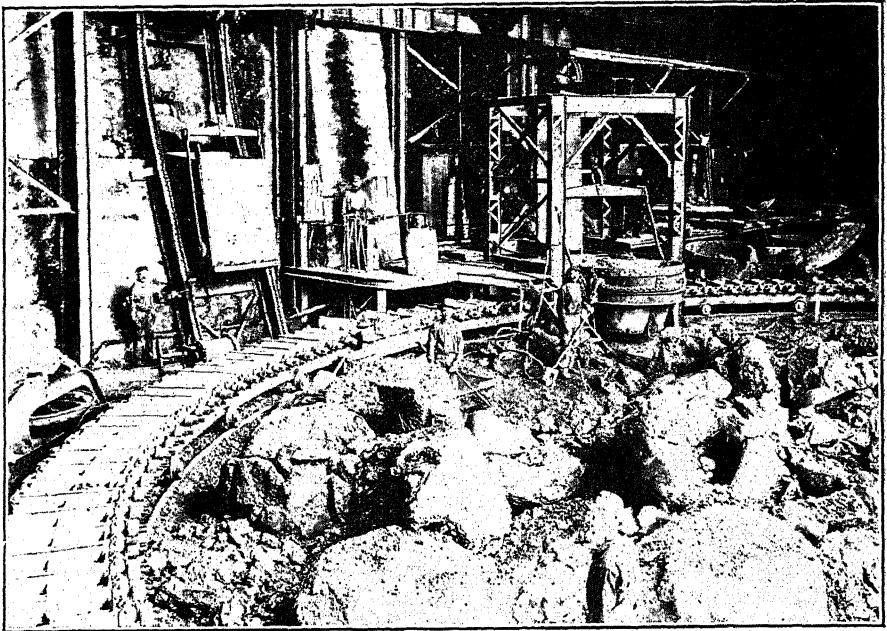
FIG. 10.—MACHINE FOR CASTING MOLTEN SLAG EQUIPPED WITH SPECIAL POURING LADLE.

the blast furnaces were started it was found that this slag was required in them as a basic flux. Accordingly the slag was poured by the crane from the ladles into large sand beds on the floor. The resulting slabs were about 6 or 7 ft. in diameter by perhaps 18 in. thick at the center, and running out to a thin edge. When these had solidified in the beds they were picked up by the crane by means of hooks, which were driven under the edge, and piled up on the floor, where they were broken up by hand labor. This was, of course, slow, laborious and expensive, and the next change was the use of a drop weight of about 4,000 lb., which was hoisted to the highest point possible by the crane and released by a trigger when a rope was pulled, thus breaking the slab. Later this weight was increased in size, and did better work than the previous one, but a great deal of hard

labor was still required to prepare the slabs for use in the furnaces. Accordingly, in 1901 designs were begun for mechanical casting of the molten slag. The machine finally built was a circular casting machine, consisting essentially of the rim of a horizontal wheel without spokes or center frame work. This wheel was 70 ft. in diameter on the center line of the molds, and was driven like a pulley by a wire cable passing around the outside of the rim and operated by two motors diametrically opposite each other. The rim of the wheel was carried on trucks running on a circular track.

The molds were carried on trunnions turning in bearings and were automatically dumped when the revolution of the whole machine had brought them to the point designed for dumping. After dumping they were automatically turned back to the proper receiving position. The center of this machine was left clear as described above, to provide space for the breaking up of the ladle skulls.

At first no special pouring ladle was provided, it being the intention to have the crane pour from one of the regular ladles directly into the molds.

This machine was started on May 23, 1902. It was found after some operation, that the pouring of slag by the crane was not satisfactory, and a special pouring ladle was installed. Fig. 10 shows the machine thus equipped and also shows the skip which is loaded from the pocket into which the molds automatically dump. This skip hoists the cast slag and dumps it into bins on the blast-furnace floor, from which the blast-furnace charge cars draw it. This machine in its original form has been in constant use since its installation and has given good satisfaction.

Many designs have been· considered for mechanically breaking and handling the ladle skulls which can be seen in Fig. 10 in the center of the machine, but none has been adopted. One of the great difficulties has been the large size of the pieces. This would require a large machine, and the tonnage to be handled is comparatively small. In the new plant now being erected, some mechanical means of doing this work will probably be adopted.

DEVELOPMENT OF THE DIFFERENT CLASSES OF CONVERTERS AFTER CLASS I.

This subject may be opened with the general statement that all con‑ verters which have been used at the plant have been of the upright type, as distinguished from the horizontal or barrel type. We have used the word "class," Class I being the earliest, Class II the next, and so on, to distinguish the different converters, and still not signify any change in the general type.

The first, or Class I, converters were circular in cross-section, 7 ft. 1.5 in. in diameter, acid lined in all except experimental runs of basic, or

neutral (chrome) lined converters, and side blast except in the bottom-blast experiments.

When the question of improving the work of the converters came prominently to the front in 1903, not from any particular occurrence, but from general observation and study of the department, the fundamental thing aimed at was to do more work, that is produce more copper, in a given

FIG. 11.—CLASS II CONVERTER, CLAY LINING.

length of time and with the same or less labor. It should be borne in mind that at this time acid linings were still in use. The accomplishment of the desired results therefore involved the following:

A converter must be made to take and finish a larger charge; therefore, it must be larger in its inside diameter when newly lined and ready for use.

It must perform the work of oxidation more rapidly, therefore air must be supplied at a faster rate, and consequently either more tuyeres or larger ones must be used.

It must make at least as many, and preferably more, charges on a

lining as the Class I, therefore it must contain more available lining material than the Class I.

It was decided that we would try to make a converter which would produce twice as much copper per charge, produce as many charges on a lining, and do this in the same aggregate time, that is, in one-half the time per ton of copper, as the Class I.

TABLE III.—*Tons Copper Produced per Lining.*

Month	Class I.	Class II.
December, 1904	15 7	22 3
January, 1905.	16 8	27 8
February, 1905	16 1	31.5
March, 1905	16 4	31.4
April, 1905	17 8	35.1
May, 1905.	19 1	. . . .
June, 1905	17.2	40 0
July, 1905	16.2	37.0
August, 1905	16 2	29.2
September, 1905	18 2	39.5
October, 1905 . . . . * . .	18.6	39 7
November, 1905	18 2	48.9
December, 1905. .	19 2	65 0 only 1 lining
January, 1906.	17 2	44 9
February, 1906.. . . .	18 0	56 7
March, 1906.	19 3	51.8
April, 1906.	16 1	46 3
May, 1906.	18 1	47 1
June, 1906	16 7	48 0
July, 1906.	16 3	37 7
August, 1906.	18 1	42 7

The first study was the distribution of the lining in the converter. It had been observed that the lining always wore away more rapidly at the tuyeres than anywhere else. The next most rapid wear seemed to be directly opposite the tuyeres, and led to the conclusion that the wear at the tuyeres was both mechanical and chemical, while that at the opposite side was largely mechanical, due to the circulation of the molten bath. The least wear appeared to be at the sides. Therefore an elliptical converter was designed with the long axis at right angles to the trunnions, that is, passing through the center of the wind box. The long diameter was 9 ft. 8 in. and the short diameter 8 ft. 8 in. The converter was not a true ellipse, but was made up of two half circles 8 ft. 8 in. in diameter, the perimeter between these being straight.

The converter is shown in Fig. 11. These figures show the shape and dimensions of the lining.

The mechanical arrangement of trunnions turning in fixed bearings was retained, as it was not considered that the weight of the converter had passed beyond the safe point for this construction. The old hydraulic tilting was, however, abandoned, and electric power substituted.

There were twelve 1⅛-in. tuyeres instead of eight ⅞-in. tuyeres, as in Class I. The height of the converter was practically the same as Class I. This was because it was not believed a greater height would be of any advantage. The converter was put into service with acid lining on Nov. 29, 1904, and its first run was very satisfactory.

The first Class II converter was operated regularly and studied as to its economy and performance as compared with the Class I converter for about 15 months before instructions were given on Apr. 5, 1906, to install the second Class II stall. This second stall was put into service on July 28, 1906, at which time we had four bowls or bottoms for the two stalls.

During the trial period preceding the starting of the second stall the work of the Class II showed rather uniform improvement as can be seen from Table III, which contains, for comparison, the performance of Class I converters during the same time.

From this it will be seen that we had succeeded in our plan to produce twice as much copper per lining as in the Class I.

As a comparison of the speed of the two classes the following can be noted:

Month, 1906	Minutes Blowing Per Ton Copper	
	Class I.	Class II.
January........	40 7	28 9
March.	40 1	24 5
July.	45 8	32 7
August.	46 9	29 8

Of course the grade of the matte affects the time required per ton of copper for blowing, but in the above comparison the two classes for any particular month are comparable as they both worked on the matte from the same sources at the same times. It will be observed that while we had not absolutely succeeded in producing copper in one-half the time per ton as in Class I, we had approached fairly near to it.

This marked the determination to abandon the Class I converters, the plan being to run each one until such time as it came to require an extensive repair, and then discard it in favor of a larger converter.

Although the abandonment of the Class I was definitely determined, the exact converter with which to replace it was not decided. A study of the work of Class II showed possibilities of improving it. For one thing, we now found that the first points at which the lining wore through were on the sides under the trunnions. We also felt that with more tuyeres we could work faster. These two things led to the design of the Class III. This is shown in Fig. 12.

It will be observed that this is an elliptical converter with the long

diameter on the line of the trunnions instead of at right angles to them as in the Class II. The long diameter is 12 ft. and the short diameter 10 ft., outside measurement. The height over all is 13 ft. 8.5 in., which is practically 1 ft. less than the Class II. There were 15 1⅛-in. tuyeres as against 12 1⅛-in. tuyeres in the Class II. It will be observed that the shape of the inside of the lining is quite different from that of the Class
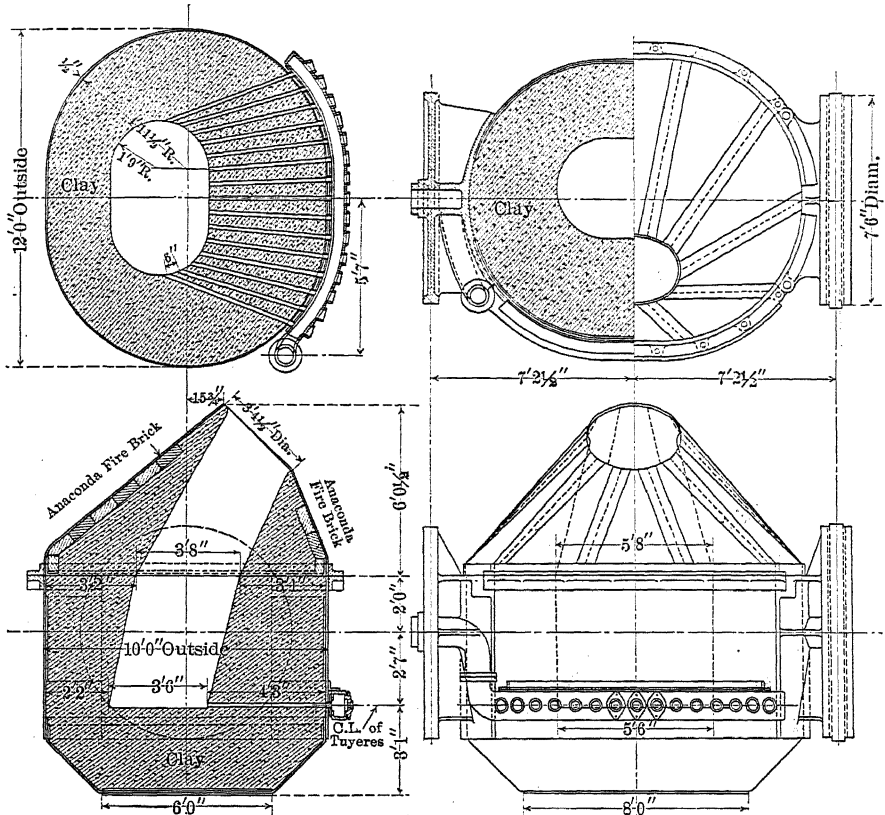
FIG. 12.—CLASS III CONVERTER, CLAY LINING.

II. This was the subject of much discussion before the adoption of the converter, as it was feared that the straight sides on the front and back would not stand up well in service. However, it was determined to try it and accordingly a new stall for this class was put in. The converter was built, and started on its first campaign on July 18, 1907, with the usual acid lining of mixed ore and B. & M. sand rock.

The first linings were put into the Class III converters by hand, as the converter would not go on the tamping machine. These runs did not show the expected increase in production per lining. It was not until March, 1908, that the converter was started on machine-tamped linings,

it having required this time to find opportunity to make the necessary changes in the tamping machine. Two hand-tamped and five machine-tamped linings were run up to the time that the teeth of the tilting gear failed on Mar. 30, 1908.

The record of the five machine-tamped linings was: Copper per lining, 70.5 tons; time blowing per ton of copper, 26.3 min.
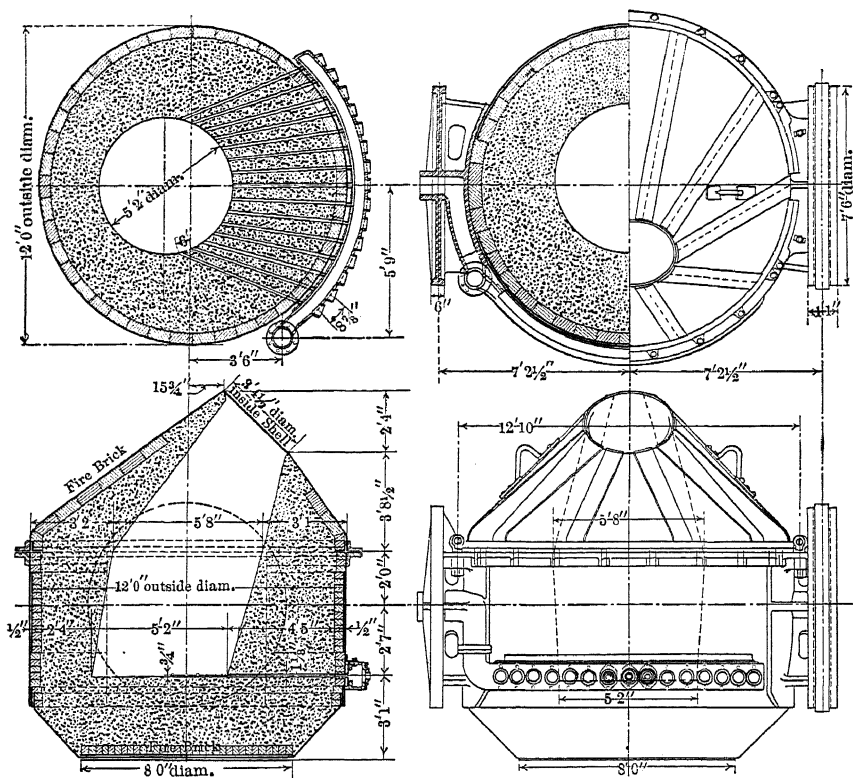
FIG. 13.—CLASS IV CONVERTER, CLAY LINING.

This was a great improvement over the work of Class II with respect to the copper produced per lining, and some improvement as regards the speed of blowing, but it is not to be considered as a fair average of the work which could be expected over a long period, as we had learned by experience with the Class II that extraordinary tonnages per lining were sometimes recorded for individual runs, due sometimes to particularly careful manipulation and sometimes to the use of a large quantity of electrolytic anode scrap which is melted up in undue proportion to the charge. Over a long period of averages this anode scrap is small in its effect on tonnages.

Study of the linings and operation of the Class III showed that the

lining wore out first at the tuyeres as in the Class I, and the next step was to retain the thickness of the Class III lining at the trunnions and thicken it up at the tuyeres and opposite them. This brought us to the Class IV, which is circular in horizontal section 12 ft. in diameter, 13 ft. 8.5 in. high (the same as Class III) with 15 1⅛-in. tuyeres (the same as Class III). This is shown in Fig. 13. The particular diameter of 12 ft. was used, as this was the same as the long diameter of the Class III and it would therefore fit in the same stall.

The date of the finished drawing for this converter is Mar. 24, 1909, and the first converter was started on Feb. 4, 1910. This class of converter was run and studied for some time with the result that on Oct. 8, 1910, instructions were given to abandon all converters except Classes III and IV as fast as they came to serious repair, it being considered that in this way the equipment would perform all converting work until basic linings which had been started should be demonstrated.

The work of seven of the early linings in Class IV converter was as follows:

Tons copper produced per lining 54.270
Tons copper produced per charge 7.448
Minutes blowing on finish per ton of copper produced 8.41

This brings us to the adoption of the basic-lined converter which will be discussed under a separate heading.

The latest development is the Class V converter but as this has been from the beginning strictly basic lined it will be discussed under that heading.

The development so far described has resulted as applied to acid-lining practice in: Increased size of individual charges; increased tonnage from one lining; decreased time blowing per ton of copper produced; decreased labor per ton of copper.

It is rather difficult to secure data which will be comparable on the time blowing for the different classes, as the grade of matte affects this item materially. In our recent studies we have adopted, as the proper comparison of the speed of the converters, the minutes blowing on the finish from white metal to copper per ton of copper produced. We call this "Minutes per ton of copper on finish." We have adopted this because no matter what has happened in converting from matte to white metal, the work to be performed per ton of copper produced from white metal to copper is the same for all charges.

We can make the general statement that from Class I to Class IV, inclusive, the speed has increased, but in the discussion of the basic-lined converter we will give some comparisons of different conditions in recent investigations.

As to labor per ton of copper produced, we can say that the Class
I converters first had crews of four men per converter. This was after-
wards cut down to three. In the later types the crew has been two men
per converter per shift. As the tonnage produced per crew shift has
increased in absolute weight, it necessarily follows that the labor per
ton has decreased considerably. The change has been made from 12-hr.
shifts to 8-hr. shifts, and the wages per man increased, but nevertheless
the cost per ton of copper for labor has decreased.

BASIC LINING.

An early attempt was made to use a basic lining and a little later a
neutral lining. Early in 1897, the exact date cannot be located, but
probably in April, one of the side-blast converters then in use, which
we now call Class I, was lined with 9-in. magnesite brick. The converter
must have been equipped with eight ¾-in. tuyeres, and was undoubtedly
in all respects, except the lining, the same as the converter shown in
Figs. 1, 2, and 3. The tuyeres were pipes, expanded into the wind box
and shell, and extending into and through the brick lining. This converter
when dried and ready was charged with molten matte, turned into the
stack and blown. As soon as the converter was blowing into the stack,
second-class ore which had been placed on a platform behind the converter,
and at a convenient height, was shoveled into the mouth. According
to the recollection of A. R. McKenzie, the converter foreman at the
time, and of which he made a record on Mar. 24, 1905, as much as 6 tons
of ore was sometimes smelted per charge, and he further states that some-
thing like 164 tons of copper was produced by the converter before the
lining was worn out.

The date is definitely fixed as not later than the early part of April,
1897, by the following note from Frank Klepetko, Superintendent at the
time, to John J. Case, Assistant Superintendent, dated Apr. 12, 1897.
"Have you had any of the ordinary slag from basic lined converter
assayed, and the silica and iron and copper determined? This should
be done regularly on each charge in order for you to get data that will
be of any value." This original note is now in the company files. Un-
fortunately a recent fire in the laboratory destroyed our early records
and these slag assays are not now available.

In February, 1901, while running a bottom-blast converter, a magnesite-
brick bottom was used as a backing for the clay bottom. This clay
bottom came up while operating the converter and therefore, in a way,
the run was partially a basic-lining run. This has been described under
the heading of "Bottom Blast".

In April, 1906, a chrome-brick lining was tried, in one of the Class I
side-blast converters such as is shown in Figs. 1, 2, and 3. A 9-in. chrome-

brick lining, backed up between the brick and the converter shell, with well-tamped clay, such as was used for acid linings, was put in with the tuyere pipes extending from the wind box into and through the brick lining.

The converter was fired for several days and when thoroughly dried was put into commission at 2 P.M., Apr. 2, 1906. The first charge was reverberatory matte assaying 36 per cent. copper. When blowing had commenced, ore from the company's Mountain View mine, Butte, was dumped into the converter. Nine charges were blown before the lining was destroyed, converting 219,225 lb. of matte and smelting or using as flux 113,309 lb. of ore. The matte probably averaged about 40 per cent. copper.

In the early part of 1910 the question of basic lining came prominently to the front, being at that time advocated by the Peirce-Smith people after their demonstrations at Baltimore and Garfield.

It was finally determined to try one of our Class IV converters without change as to shape and mechanical arrangements, with a basic lining of magnesite brick.

The converter, which was known as bottom B, was lined as shown in Fig. 14, and the first matte charged into it at 3.50 P.M., Mar. 9, 1911. It had 15 tuyeres of 1.25-in. extra strong tubing having an internal cross-section of 1.27 sq. in. The converter started off well and continued to run well, so that on Apr. 29, 1911, the second converter was put into service. This was an exact duplicate of the first converter and was known as Class IV, bottom C. The third converter, a duplicate of the first two, and known as Class IV, bottom D, was put into service on May 7, 1911.

In the meantime the question of the number and size of tuyeres had been discussed with the result that the fourth converter known as Class IV, bottom A, was a duplicate of the first three except that it had 26 tuyeres, each with an internal cross-sectional area of 1.27 sq. in. This was put into service June 22, 1911.

We then considered that we would need a spare converter and therefore lined a Class III bottom, putting in 24 tuyeres, each with 1.27 sq. in. internal cross-section.

These are all the bottoms which have been lined, except the Class V, 20-ft. converter. Some of them have finished the first campaign, been repaired and started on the second. Some were shut down before completing a campaign and changes made for experimental purposes, and some are still running on the first campaign.

A brief résumé of each lining is as follows:

(A tabulation of these figures with some calculations from them will be found in Table IV.)

Class IV, Bottom B.—Started with 15 tuyeres each with 1.27 sq. in. internal cross-section; first charge, Mar. 9, 1911; ran until June 3, 1912, an elapsed time of 451 days; made 1,026 charges, producing 12,216.1 tons of copper; the calculated matte treated is 35,700 tons; ore smelted,
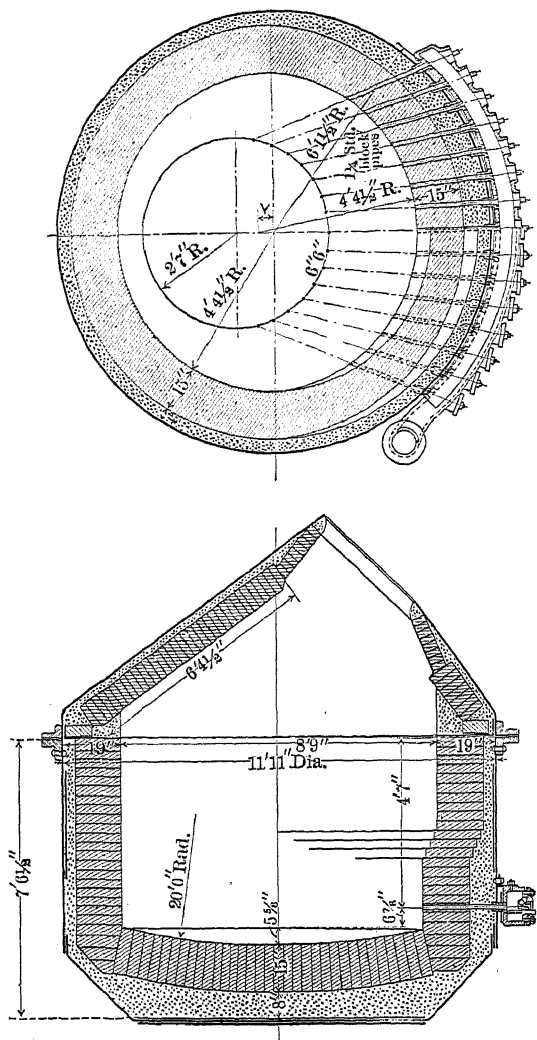
FIG. 14.—CONVERTER B, BASIC LINING.

8,398.8 tons; time operating, 7,509 hr. 45 min.; shut down to repair lining.

Class IV, Bottom C.—Started with 15 tuyeres, each having 1.27 sq. in. internal cross-section; first charge, Apr. 29, 1911; ran until Jan. 4, 1912,

an elapsed time of 251 days; made 580 charges, producing 7,479 .1 tons of copper; the calculated matte treated is 23,500 tons; ore smelted, 5,222.8 tons; time operating, 4,149 hr. 40 min. The converter needed only a slight repair when it was shut down but was shut down in order to put in 34 tuyeres for experimental purposes. This converter was then fitted with 34 cast tuyeres, as described under the heading "Cast Copper Tuyeres."

It was put into service with these cast tuyeres Mar. 17, 1912, and run on them experimentally until Mar. 27, 1912, when the tuyeres burned out. These cast tuyeres were then replaced with 34 tuyeres, each having an internal cross-section of 1.27 sq. in. The converter was run on these tuyeres for a few days, during which it threw out its charge badly and therefore instructions were given to plug the two end tuyeres on each end, leaving 26 tuyeres in action. Therefore practically all of this second run was made on 26 1 .25-in. tuyeres, and this will be considered as its tuyere equipment.

Its record on this run is as follows:—Charged Mar. 17, 1912; up to Apr. 13, 1913, elapsed time 392 days; made 445 charges producing 5,643 tons copper; the calculated matte treated is 16,900 tons; ore smelted, 3,176 tons; time operating 2,960 hr. 10 min.; still in service, after some slight repairs to lining.

Class IV, Bottom D.—Started with 15 tuyeres, each having 1.27 sq. in. internal cross-section; first charge, May 7, 1911; ran until Dec. 22, 1912, when it was shut down to change the tuyeres for experimental purposes; elapsed time up to Dec. 22, 1912, 596 days; made 1,014 charges, producing 13,414 tons copper; calculated matte treated, 40,250 tons; ore smelted, 10,157 tons; time operating, 7,663 hr. 55 min.

The converter was shut down to try an experiment with large tuyeres, and was equipped with 15 tuyeres of 3-in. boiler tube with an approximate inside diameter of 2 .75 in. and an area of 6 .079 sq. in. each.

The converter was started on Dec. 28, 1912, but only seven of the tuyeres were used, the others being plugged with clay. The converter ran with these tuyeres until Apr. 2, 1913, when it was decided that the tuyeres were too large. The elapsed time was 96 days, in which 85 charges were made, producing 968 .2 tons of copper. The estimated matte treated is 2,900 tons; ore used, 816 tons; time operating, 601 hr. 45 min.

When the converter was shut down it was equipped with 15 tuyeres of 2 .5-in. boiler tubes with an internal area of 4 .09 sq. in. each, and started again on May 12, 1913, and is still in operation.

During this time of May 12 to June 1, it has made 28 charges, producing 369 .3 tons of copper; estimated matte treated, 1,110 tons; ore used, 294 tons; time operating, 184 hr.

All of these three runs can really be considered as one campaign, as in

no case was the converter shut down from necessity to repair. The shut-downs were all for purposes of changing the tuyeres for experimental purposes.

Considering these as one campaign, the record is:—Started May 7, 1911; running at present, the elapsed time to June 1, 1913, being 756 days, although a few days were lost in making tuyere changes; made 1,127 charges producing 14,751.5 tons copper; estimated matte treated, 44,260 tons; ore used, 11,267 tons; time operating, 8,449 hr. 40 min.

Class IV, Bottom A.—This was the first trial of an increased number of tuyeres. The converter was equipped when it started with 26 tuyeres, each having 1.27 sq. in. internal cross-section. It was started June 22, 1911, and is still operating, the elapsed time up to June 1, 1913, being 722 days; made 1,170 charges, producing 16,407.3 tons of copper; estimated matte treated, 49,200 tons; ore used, 10.118 tons; time operating, 8,198 hr. 40 min.

Class III, Bottom 3.—This was lined up as a spare bottom and there-fore has not been operated continuously, which should be borne in mind in considering its record. It was first equipped with 24 tuyeres, each having 1.27 sq. in. internal cross-section. It was started Nov. 16, 1911, and continued with this tuyere equipment until Dec. 25, 1912, when it was shut down to change tuyeres for experimental purposes. The elapsed time to Dec. 25, 1912, is 406 days, in which 640 charges were made, pro-ducing 8,990.7 tons of copper; estimated matte treated 27,000 tons; ore used, 5,171 tons; time operating, 4,523 hr. 50 min.

When the converter was shut down it was for the purpose of equipping it with 11 tuyeres of 2.5-in. boiler tube, each tuyere with an internal cross-section of 4.09 sq. in. It was started Mar. 28, 1913, and is still running, the elapsed time to June 1, 1913, being 70 days. In this time 33 charges were made, producing 404.2 tons of copper; estimated matte treated, 1,210 tons; ore used, 389 tons; time operating, 251 hr. 50 min.

Here again the two runs can really be considered as one campaign, and if they are combined the record is:—Started Nov. 16, 1911, still running, the elapsed time up to June 1, 1913, being 476 days; made 676 charges producing 9,394.9 tons copper; estimated matte treated, 28,210 tons; ore used, 5,560 tons; time operating, 4,775 hr. 40 min.

Class V, or 20-ft. Converter.

All our investigations and experiments pointed to the probable success of a larger converter than the Class IV or 12 ft. After some discussion, it was determined to build one of about three times the capacity of the Class IV and 20 ft. was decided upon as the diameter, still adhering to the up-right type.

The design involved many mechanical problems, but it is probably sufficient to say that the converter as built has shown no structural weakness in its operation to date of 269 days.

One of the important discussions was as to the number and size of tuyeres to be put in it. It was finally decided to equip it with 62 tuyeres
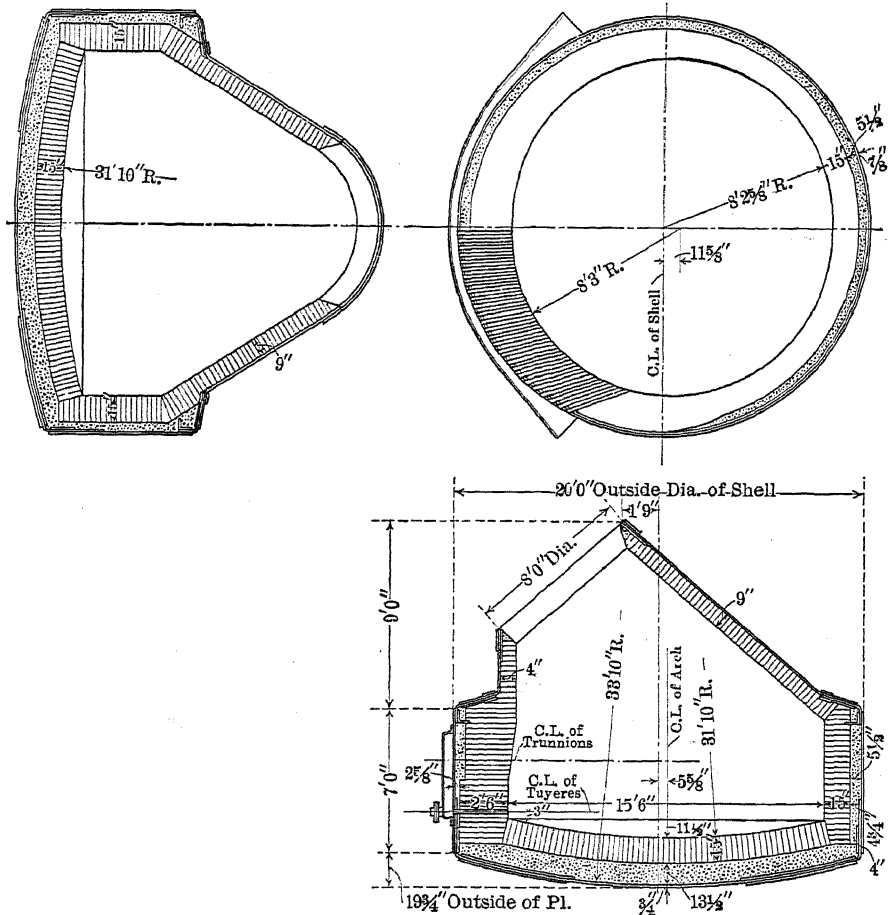
FIG. 15.—20-FT. CONVERTER, BRICK LINING.

of 2-in. boiler tube, each having an inside diameter of approximately 1.75 in. and a transverse internal area of 2.573 sq. in. The reason for this decision is related under the heading "Number and Size of Tuyeres."

The shape and principal dimensions both inside and outside are shown in Figs. 15, 16 and 17.

After running a time it was decided to change the tuyeres to a less number of a greater diameter, which will show in the following record:

Class V, Bottom A.—Started Aug. 3, 1912; run until Dec. 17, 1912, with 62 tuyeres, each with an internal cross-section of 2.573 sq. in.; elapsed time 137 days; made 88 charges, producing 2,835 tons copper; estimated matte treated 8,500 tons; ore used 1,368 tons; time operating 656 hr. 45 min.

On Dec. 17, 1912, it was shut down to replace the 62 1.75-in. tuyeres with 26 2.25-in. tuyeres, made from 2.5-in. boiler tube, and having an internal cross-section of 4.09 sq. in. each. Started with 26 tuyeres Jan. 30, 1913; still operating, the elapsed time to June 1, 1913, being 132
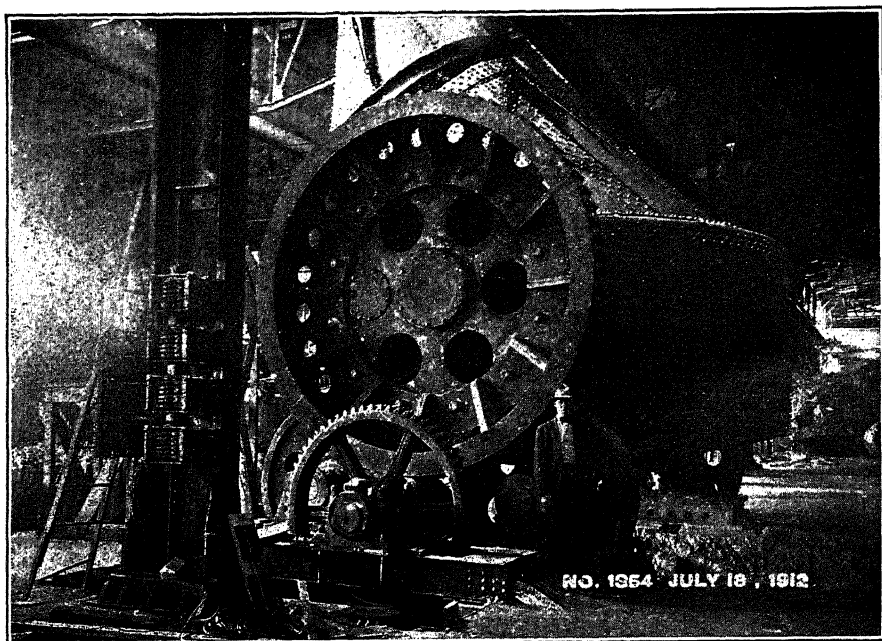
FIG. 16.—CLASS V CONVERTER. SIDE VIEW.

days; made 170 charges, producing 6,223.2 tons copper; estimated matte treated, 18,700 tons; ore used, 3,982 tons; time operating, 1,306 hr.

For the purpose of bringing the above record of the different converters together in more concise form Table IV is given, and from this tabulation another will be made bearing upon the determination of the best size and number of tuyeres.

The total time operating is the summation of the total elapsed time from charge to finish of all the individual charges and therefore includes all the usual delays of daily operation.

The copper per converter-hour is the total copper produced divided by the operating time.

It is quickly seen that the Class V converter greatly outstrips all others

TABLE IV.—*Record of Basic-Lined Converters.*

Converter		Campaign		Elapsed Days	Tons Cu	Charges	Tons Cu Per Charge	Total Time Operating		Tons Cu Per Converter-Hour	Tuyeres	
Class	Bottom	From	To					Hr.	Min.		No.	Approx. Internal Diam. In.
IV	B	Mar. 9, 1911	June 3, 1912	451	12,216.1	1,026	11.9	7,509	45	1 63	15	1 25
IV	C	Apr. 29, 1911	Jan. 4, 1912	251	7,479.1	580	12.9	4,149	40	1 80	15	1 25
aIV	D	May 7, 1911	Dec. 22, 1912	596	13,414	1,014	13.2	7,663	55	1 75	15	1 25
bIV	A	June 22, 1911	June 1, 1913	710	16,407.3	1,170	14.2	8,198	40	2 00	26	1 25
cIII	3	Nov. 16, 1911	Dec. 25, 1912	406	8,990.7	640	14.0	4,523	50	1 99	24	1 25
IV	C	Mar. 17, 1912	Apr. 13, 1913	392	5,643	445	12.7	2,960	10	1 91	34	1 25
V	A	Aug. 3, 1912	Dec. 17, 1912	137	2,835	88 1/6	32.4	656	45	4 31	62	1 75
bIV	B	Sept. 12, 1912	June 2, 1913	......	3,904.3	307	12.7	2,180	20	1 79	26	1 375
aIV	D	Dec. 28, 1912	Apr. 1, 1913	96	968.2	85	11.4	601	45	4 77	7	2 75
bV	A	Jan. 20, 1913	June 1, 1913	132	6,223.2	109.9	36.6	1,306	0	1 60	26	2 25
cIII	3	Mar. 28, 1913	June 1, 1913b	70	404.2	33	12.3	251	50	1 60	11	2 25
aIV	D	May 12, 1913	June 1, 1913	20	369.3	28	13.2	184	0	2 01	15	2 25

a These three runs are really parts of the same campaign.
b Still running on this campaign.
c These two runs are really parts of the same campaign. The converter was shut down on Dec. 25, 1912, to change the tuyeres for experimental purposes.

as regards speed of operation, that is copper per converter-hour, and also as to copper per charge.

The comparison of the operating labor on the 12-ft., or Class IV, and the 20-ft., or Class V, is as follows, each of course operating three shifts per 24 hr.:

Labor	Rate Per Shift	12-Ft. Converter				20-Ft. Converter			
		Per Shift		Per Day		Per Shift		Per Day	
		No.	Cost	No.	Cost	No.	Cost	No.	Cost
Skimmer...	$4.25	1	$4.25	3	$12 75	1	$4.25	3	$12 75
Helper. ...	3 25	1	3 25	3	9.75	2	6.50	6	19.50
Total ...	...	2	$7 50	6	$22.50	3	$10 75	9	$32.25

When it is considered that the 20-ft. converter produces three times the copper per day that the 12-ft. converter does, the comparison becomes: Labor cost one 20-ft. converter, $32.25; labor cost three 12-ft. converters, $67.50.

NUMBER AND SIZE OF TUYERES.

In order to segregate this important subject a rearrangement of parts of the tabulation "Performance of Basic-Lined Converters" is given in the accompanying table in which the performance of the different converters is listed in approximately the order of increasing total tuyere area.

TABLE V.—*Converter Performance with Relation to Number and Size of Tuyeres.*

Converter		No.	Tuyeres		Total Area Sq. In.	Tons Copper Produced		
Class	Bottom		Approx. Diam. In.	Internal Area Sq. In.		Per Charge	Per Converter-Hour	Total for Campaign
IV	B	15	1.25	1 27	19.05	11.9	1.63	12,216.1
IV	C	15	1.25	1 27	19.05	12.9	1 80	7,479 1
IV	D	15	1.25	1.27	19 05	13.2	1 75	13,414.0
III	3	24	1.25	1 27	30.48	14.0	1.99	8,990.7
IV	A	26	1.25	1.27	33 02	14.2	2 00	16,407.3
IV	C	26	1.25	1.27	33.02	12.7	1.91	5,643 0
IV	B	26	1.375	1.496	38.896	12.7	1 79	3,904.3
III	3	11	2.25	4 09	44.99	12.3	1.60	404.2
IV	D	7	2.75	6.079	42.553	11 4	1.69	968.2
IV	D	15	2.25	4 09	61.35	13.2	2.01	369.2
V	A	62	1.75	2.537	157.294	32.4	4.31	2,835
V	A	26	2.25	4.09	106.34	36.6	4.77	6,223.2

An inspection of the tabulation shows that the first increase from 15 to 24 tuyeres of the same size resulted in faster work of the converter. The increase to 26 tuyeres showed practically the same work as with 24. The 26 1.375-in. tuyeres show a little falling off which is not explained.

The Class IV, bottom B, with 26 1.375-in. tuyeres shows a little falling off in its speed as compared with converters with 26 1.25-in. tuyeres and having therefore a little less total tuyere area. This is because it
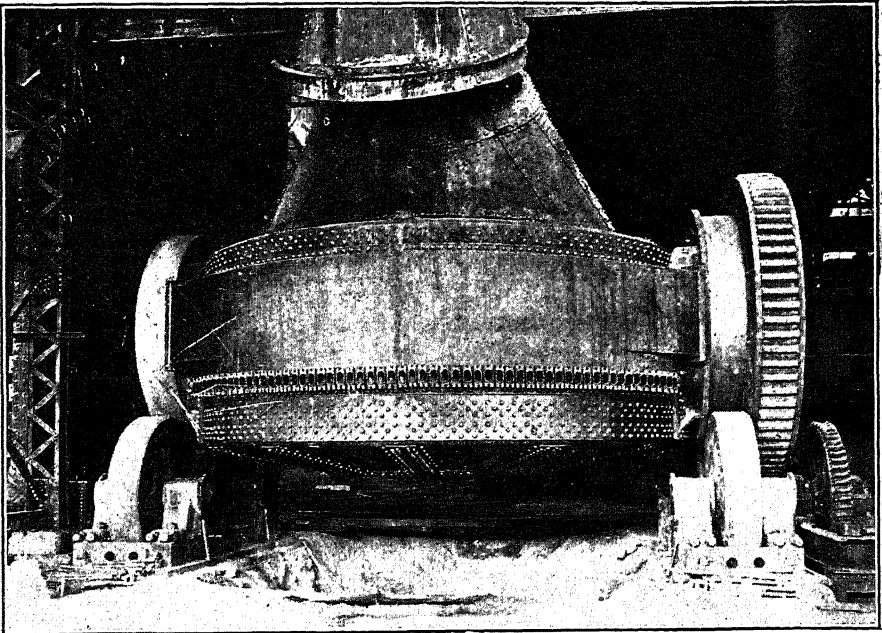
FIG. 17.—CLASS V CONVERTER. FRONT VIEW.

was operated a large portion of the time in a stall having many bends in its air-supply pipes, resulting in restriction of the air supply. Any converter used in the same stall has shown this same tendency as compared with the same converter in other stalls.

The Class III converter with 11 2.25-in. tuyeres shows a falling off in its speed, but this is explained by the fact that it was used intermittently and in the stall with many bends in the air pipe, as described above for Class IV, bottom B.

When we come to the bottom D with seven 2.75-in. tuyeres (3-in. boiler tube) we find a falling off. Observations on this converter showed a great tendency for it to throw out its charge, due evidently to the concentration of air in large volume in a few places.

This same tendency was shown by bottom C with 34 1.25-in. cast

tuyeres, and was probably due to the concentration of the air due to close spacing of the tuyeres.

When we come to bottom D with 15 2.25-in. tuyeres we find work equally as good as the bottom A with 26 1.25-in. tuyeres and the fewer tuyeres of larger size are much preferable for construction reasons, for the strength of the lining, and for the labor of punching.

The Class V converter shows the same thing, the 26 2.25-in. tuyeres showing even better work than the 62 1.75-in. tuyeres, and they are far preferable from all points of view.

Another point of superiority of a few large tuyeres, within the limits of working, over many small ones, is in the decreased resistance to the air and consequently a lower necessary blast pressure. We have found this to be true from practical observation as well as from theoretical reasoning, but have not yet carried our experiments far enough to include the results in this paper.

In order for the increase in tuyere area to be truly of the benefit the preceding figures indicate, it must be shown that the air when passing through is used economically. That it is used at as high efficiency at large volume as when used at less volumes per minute will be shown in detail under "Efficiency of Air."

Cast-Copper Tuyeres.

In the history of Class IV, bottom B, basic-lined converter, mention was made of cast-copper tuyeres. The following discusses this arrangement a little more in detail.

There is some difficulty in fitting the magnesite brick of the lining around the circular tuyere pipes, and where tuyeres are closely spaced the brick work between them is so small that it is apt not to stay in place well.

With the idea of simplifying the construction and also of allowing the close spacing of tuyeres a Class IV magnesite-lined converter was equipped with 34 tuyeres 1.27-in. inside diameter. Each tuyere was a cast-copper block of such a shape and dimension that it would lie flat on the brick work, which was built up to just below the tuyere line, and fill the space horizontally from half way between itself and the tuyere on the right, and itself and half way between the tuyere on the left, so that when all tuyere blocks were in place they formed a solid block with a tuyere opening through each one. They were made by laying a copper pipe with an inside diameter equal to the proposed inside tuyere diameter, in a dry sand mold and casting the copper around it, allowing the pipe to extend out of the end of the block so as to be expanded into the converter shell like any ordinary tuyere pipe.

The converter was tried over the period Mar. 15 to 27, 1912, with 29 different charges, some of which were blown only from white metal to copper, some from matte to white metal, and some were not finished to either.

Without going into details it can be said that it was found that the tuyeres would stand fairly well while blowing from matte to white metal, but that they melted away rapidly when blowing from white metal to copper. The converter threw out badly, due possibly to the concentration of air in large volumes at particular places because of the ring being melted back.

It was thought when the plan was first discussed that the air blast passing through the blocks would keep them sufficiently cool to prevent melting, but it evidently did not. The plan was abandoned as not holding out sufficient promise of success to warrant continued experiments.

EFFICIENCY OF AIR.

The speed with which a converter will work, that is, perform oxidation, is of course dependent upon the rapidity with which the chemical reactions will take place and the rate at which air, or, strictly speaking, oxygen, is supplied. With proper distribution of the air through the charge the speed of the reactions in the converter is undoubtedly beyond any practical rate of supplying the air. It is therefore of interest and of practical importance to know how fast air can be supplied and still be efficiently used, and how the air must be distributed.

Our first experiment along this line was on the distribution of the air. A light tin tank about 20-in. in diameter and 2 ft. 6 in. high, with a glass bottom, was made. Two wind boxes were made independent of the tank so that they could be attached to an air main and placed into the tank in the same relative position as a wind box on a converter.

One of these wind boxes had eight holes, each $\frac{1}{4}$ in. in diameter, and the other had 32 holes $\frac{1}{8}$ in. in diameter. Thus the total area of the holes was the same in both cases.

When the tank was partly filled with water and the eight-hole wind box tried, it was found that the air penetrated the water only a few inches, rising immediately to the surface in comparatively large bubbles. When the 32-hole wind box was tried the air acted in the same way, except that the bubbles coming to the surface were much smaller and better disseminated through the liquid. Incidentally it may be mentioned that the circulation of the charge as represented by the water was upward from the tuyeres, the surface then flowing away from the tuyeres and toward the front, thence downward toward the bottom, thence across the bottom toward the tuyeres. There was not sufficient penetration of the blast into the charge along the bottom to set up a circulation in the

direction opposite to that described as had been thought probable. After several observations had been made, all showing the same results, it was decided that a large number of small tuyeres would probably result in more efficient use of the air than fewer large tuyeres.

These experiments were conducted to help in making the decision as to the number of tuyeres which should be put into the Class V or 20-ft. converter which was then being built, and as a result, it was decided to place the tuyeres as close together as construction details would allow, thus providing as much distribution of air as possible. They were made as large as it was felt would work to advantage, so as to make the converter work as fast as possible. Consideration was given to the probable increased resistance to the passage of the air through many small tuyeres as compared with few large ones, but air distribution was given the preference, and the first Class V converter was equipped with 62 tuyeres of 2-in. boiler tube with an approximate inside diameter of 1.75 in. As described in the discussion of the Class V converter, these were afterwards changed to 26 tuyeres of 2.5-in. boiler tube with an approximate internal diameter of 2.25 in.

After this converter was in operation it was desired to know whether the large volume of air put into the converter was efficiently used and therefore experiments were outlined and carried out for this purpose.

The method adopted for the determination of the efficiency of the air was a chemical one based on the fact that the nitrogen entering the tuyeres passed out of the mouth of the converter unchanged chemically. Therefore, if the ratio between the oxygen and nitrogen in the air, and the same ratio in the outgoing gases be known, the nitrogen being a fixed quantity, the percentage of the entering oxygen which is used, or the efficiency as we call it, can be calculated from the formula:

$$\frac{\frac{O^1}{N^1} - \frac{O}{N}}{\frac{O^1}{N^1}} \times 100 = \text{per cent. efficiency.}$$

Where O^1 = oxygen in the air; N^1 = nitrogen in the air; O = oxygen in escaping gases; N = nitrogen in escaping gases.

This necessitates the sampling of the gases actually issuing from the mouth of the converter before any admixture of atmospheric air with them. This was done by putting the sampling pipe into the mouth of the converter.

The first samples were taken through an iron pipe the inside of which was washed with milk of lime to prevent oxidation of the iron and consequent vitiation of the sample. It did not prove satisfactory as no oxygen could be found in any of the samples of escaping gases, or in other words, the samples always showed 100 per cent. efficiency. It did not

seem reasonable to suppose that such uniformly perfect results should be obtained, and it was concluded that the oxygen in the gases was used in oxidizing the hot walls of the pipe.

The final sampling pipe was of iron, lined with porcelain tubes. The end of the iron pipe which was inserted in the converter was protected by covering the outside with a clay coating.

The samples were all taken over mercury by aspiration, the acid gases, that is the SO_2 and SO_3, absorbed by KOH, the oxygen absorbed by pyrogallic acid and the nitrogen determined by difference. Results quoted here are only for samples taken with the proper sampling tube.

For the purpose of comparison samples were also taken of the gases from a Class IV or 12-ft. converter having 26 1.25-in. tuyeres. The results are shown in Table VI.

An examination of the results shows almost uniformly high efficiencies and a study of the figures is of interest.

Picking out all the 100 per cent. efficiencies we find volumes in cubic feet per minute as follows: 16,700, 20,800, 20,000, 24,000, 25,000, and 32,200, which would show that air in volumes up to 25,000 in cubic feet per minute, and even up to 32,200 cu. ft., can be used efficiently and the readings quoted are all on the 20-ft. converter. On the other hand we find a reading of 24,400 cu. ft. with an efficiency of only 83.3 per cent. This was near the finishing of the charge, when it may be possible that air will not be used so efficiently on account of the absence of iron and the small quantity of sulphur present, and available for reaction.

The high as well as the low efficiencies are found with a wide range of volumes which would indicate that conditions may vary and affect the efficiency, but that it is entirely practical and commercial to use large volumes of air, and this is the principle which has been adopted in the design of converters for the plant.

Size of Converter Mouth.

Considerable experimenting on this question has been done, and without going into an extended discussion of the details it may be stated that a large mouth is an advantage. The largest which has been tried on the Class IV converter is a 6 ft. 6 in. diameter opening in the shell. On the Class V converter this opening is 8 ft. diameter.

The advantages of this are: It is easier to keep clean, that is to remove the crusts; less tendency to close up; easier to put in the charge, ore, etc.

The closing up of the mouth by the building up of crusts will result in making it difficult for the.waste gases to escape freely with the result that the converter will become hot and injure the lining. On the other

TABLE VI.—Efficiency of Air in Converting.

Date 1912	Converter Class	Converter Bottom	Converter Diameter Ft.	Time	Sample No.	Tuyers No.	Tuyers App. Inter. Diameter In.	Cu Ft. Air Per Min. Stand. Conditions	Acid Gas	O	N	Pct. Cent. Efficiency of Air	Remarks
Aug. 14	V	A	20	10.00 a.m.	1	62	1¾	22,400	28.9	2.7	68.4	85.2	Sample taken after second doubling of charge. Strong SO₂ odor.
				11.10 a.m.	2	62	1¾	24,800	10.6	4.8	84.6	78.3	After charging 1 boat of ore and 1 pot of matte.
Aug. 15	V	A	20	11.45 a.m.	1	62	1¾	18,000	13.6	1.4	85.0	94.8	Converter charged at 8 a.m. No punching.
				1.45 p.m.	2	62	1¾	18,000	13.4	1.3	85.3	94.1	
				2.45 p.m.	3	62	1¾	24,400	20.7	3.3	76.0	83.3	Converter about 1 hr. 30 min. on finish.
Aug. 16	V	A	20	9.45 a.m.	1	62	1¾	15,740	18.0	2.4	79.6	88.4	5 min. after charging 1 boat of ore.
				1.25 p.m.	2	62	1¾	16,700	19.9	0.0	80.1	100.0	5 min. after charging 1 boat of ore and 2 pots of matte.
				2.30 p.m.	3	62	1¾	20,800	20.8	0.0	79.2	100.0	
				3.10 p.m.	4	62	1¾	25,200	16.8	0.5	82.7	97.8	
Aug. 17	V	A	20	9.45 a.m.	1	62	1¾	19,200	7.8	0.5	91.7	97.8	5 min. after charging 1 boat of ore and 1 pot of matte.
				10.45 a.m.	2	62	1¾	20,000	49.3	0.0	50.7	100.0	
				1.25 p.m.	3	62	1¾	16,850	27.3	0.5	72.2	97.6	5 min. after charging 1 pot of matte.

Date				Time									Remarks
Aug. 17	V	A	20	2.10 p.m.	4	62	1¾	22,800	14.3	0.9	84.8	97.0	5 min. after start on finish. Charge very cool. No punching.
				3.15 p.m.	5	62	1¼	20,000	6.6	1.4	92.0	94.2	
Aug. 19	IV	A	12	11.05 a.m.	1	26	1¼	7,200	8.0	3.2	88.8	86.4	5 min. after charging 1 pot of matte.
				11.45 a.m.	2	26	1¼	9,800	10.8	0.6	88.6	97.6	On finish 20 min.
				12.40 p.m.	3	26	1¼	9,700	14.7	0.6	84.7	97.5	On finish 1 hr. 20 min.
				2.15 p.m.	4	26	1¼	6,400	7.8	0.4	91.8	98.5	5 min. after charging 1 boat of ore.
				2.45 p.m.	5	26	1¼	8,400	13.8	0.4	85.8	98.2	Doubled with 2 pots of matte at 3.30 p.m.
				3.40 p.m.	6	26	1¼	6,800	10.5	0.6	88.9	97.5	
Aug. 20	V	A	20	9.20 a.m.	1	62	1¾	19,500	14.0	0.7	85.3	97.0	Blown in at 9 a.m. with 6½ pots inside matte, and 1¾ pots trans.
				10.05 a.m.	2	62	1¾	24,000	13.8	0.0	86.2	100.0	After skimming ½ pot of slag, and charging 1 boat of ore.
	IV	A	12	1.15 p.m.	3	26	1¼	11,500	12.6	1.0	86.4	95.4	On finish, 3 min. after charging 1 boat of ore. Punching.
				1.55 p.m.	4	26	1¼	10,700	15.4	0.4	84.2	98.2	2 min. after charging 1 boat of scrap.
				2.20 p.m.	5	26	1¼	8,850	16.9	0.1	83.0	99.7	
				3.40 p.m.	6	26	1¼	8,750	9.6	0.4	90.0	98.2	3 min. after charging 1 boat of ore and 2 pots of matte.
				4.15 p.m.	7	26	1¼	7,900	10.0	0.9	89.1	95.9	10 min. after charging 1 boat of ore.
Aug. 21	V	A	20	10.40 a.m.	1	62	1¾	19,400	10.8	0.6	88.6	97.4	Charged at 7.35 a.m., doubled at 10.15 a.m.
				11.30 a.m.	2	62	1¾	23,800	16.8	0.3	82.9	98.5	5 min. after charging 1 pot of matte and 1 boat of ore.

TABLE VI.—*Efficiency of Air in Converting (Continued).*

Date 1912	Converter			Time	Sample No.	Tuyeres		Cu. Ft. Air Per Min. Stand. Conditions	Volumetric Analysis of Gases			Per Cent. Efficiency of Air	Remarks
	Class	Bottom	Diameter Ft.			No.	App. Inter. Diameter In.		Acid Gas	O	N		
Aug. 21	V	A	20	1.25 p.m.	3	62	1¾	22,400	13 4	0 6	86.0	97 4	Charged 1 boat of scrap.
				2.05 p.m.	4	62	1¾	31,400	14 6	0 2	85 2	98 2	Charged 1 boat of scrap.
				2.40 p.m.	5	62	1¾	23,600	33 1	0 1	66 8	94 8	
				3.15 p.m.	6	62	1¾	23,200	35 6	0 1	64 3	94 0	
Aug. 22	V	A	20	8.15 a.m.	1	62	1¾	25,000	10 9	0 0	89 1	100 0	Converter charged at 8 a.m.
				9.25 a.m.	2	62	1¾	19,500	9.9	0 1	90.0	99 8	5 min. after charging 3 pots matte, 1 boat ore, and 1 boat scrap.
				10.10 a.m.	3	62	1¾	18,700	10 1	0 4	89.5	98 2	5 min. after charging 1 pot matte.
				10.55 a.m.	4	62	1¾	22,800	9 2	0 2	90 6	99 3	
				11.35 a.m.	5	62	1¾	22,800	11 1	0 4	88.5	98 2	
				12.30 p.m.	6	62	1¾	21,800	10 6	0 3	89 1	98 8	
				1.35 p.m.	7	62	1¾	25,400	22.7	0 0	77.3	100 0	Charged ½ large pot matte. 5 min. after charging pot of matte.
				2.35 p.m.	8	62	1¾	31,800	24 4	0.8	74 8	96 0	On finish 1 hr. 45 min.
				3.00 p.m.	9	62	1¾	32,200	21.5	0.0	78 5	100 0	Charged 1 boat of dope.

hand the mouth can be made so large as to allow too free an escape of the gases with the consequent too rapid escape of heat, and a decrease in the cold material which can be smelted.

HEIGHT OF CONVERTER.

One of the Class I converters was changed by inserting an extension ring 19.75 in. high between the main trunnion section and the tuyere section, and then putting on a tall cap, making the converter 18 ft. 2.25
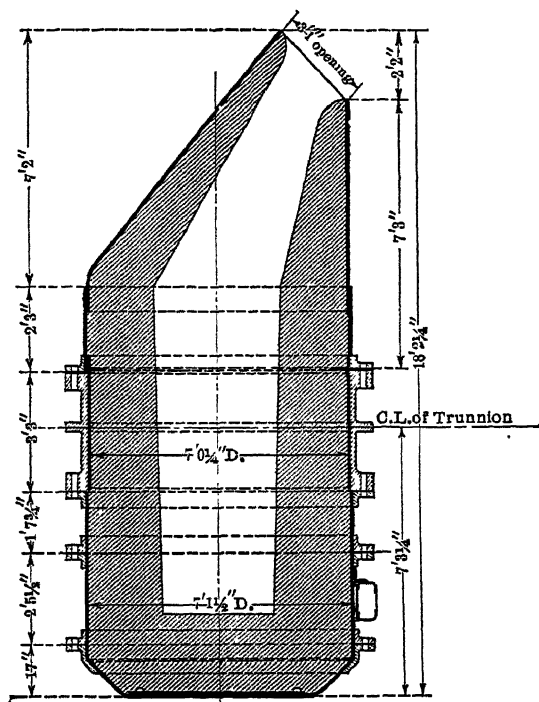
FIG. 18.—CLASS I CONVERTER, LENGTHENED 4 FT.

in. high over all and 13 ft. 5.75 in. from the center of the tuyeres to the lowest point of the mouth. A vertical section of this converter is shown in Fig. 18.

This was done to try its effect as to the throwing out of the charge by the blast. After rather an extended trial, the details of which will not be given here, it was determined that no advantage was obtained. These trials were made in August, 1910.

In view of the fact that the Class IV converter 13 ft. 8.5 in. high, and the Class V converter 17 ft. 7.75 in. high do not throw out seriously, it would seem that no advantage would be gained by making the converter

higher. In fact there is a distinct disadvantage as was demonstrated in the trials of the tall converter, namely the clogging up of the mouth by the charge throwing against the lining in the upper part of the cap and chilling there.

Summary.

The basic lining has shown its superiority over the acid lining and therefore the acid lining can be dismissed as not being the best practice.

A large converter is better than a small one. There is no practical limit to the size of a converter which will work metallurgically. The limits are mechanical, and depend on the size of unit a plant will stand from the point of view of quantity of production, but even in this latter instance there can be a rather wide range of choice. We have found that the Class V or 20-ft. converter can be used as a storage for matte or for finished copper for long periods if the mouth is properly covered. Therefore in a plant of comparatively small production, a combination of ample matte storage capacity in blast-furnace settlers and reverberatory furnaces, together with a large converter, can be made so as to allow all converting to be done on one shift, and the casting of the copper perhaps to be done on the next shift.

With the success of the large units even the largest plants should have only a few large units and not a large number of small units.

From the experiments and observations which have been made, we feel that the tuyeres for any converter from 12 to 20 ft. diameter should be about 2.25 in. internal diameter. Tuyeres 2.75 in. internal diameter have not proved successful. Tuyeres smaller than 2.25 in. diameter introduce undesirable construction features, particularly weakening the lining at the tuyeres, introduce unnecessary resistance to the entrance of the air, and add to the work of punching.

The mouth of the converter should be of ample size. Openings of 6 ft. 6 in. diameter for a 12-ft. converter and 8 ft. for a 20-ft. converter have not been found too large.

The bottom lining must not be too close to the tuyeres on account of a tendency of the bottom to build up and interfere with the punching of the tuyeres and also often causing a concentration of air in a few places with consequent throwing out of the charge. At least 5 in. should be allowed from the lowest point of the tuyeres to the bottom lining. There is no reason why this dimension cannot be increased, the determining factor being the size of charge it is desired to finish. There must always be copper enough at the finish to cover the tuyeres and a very deep bottom would require a large charge.

Air can be used efficiently in large volumes, for instance in the 20-ft. converter up to 22,000, or even more, cubic feet per minute. However,

the best results seem to be for this converter about 18,000 cu. ft. while slagging.

As to the height of the converter, the Class IV, 13 ft. 8.5 in. high, and the Class V, 17 ft. 7.75 in. high, have proved very satisfactory with proper attention to the details of operation, and trials of a taller converter were not satisfactory. Probably the converter should not be made appreciably lower. The proper dimension to quote in this particular is from the center of the tuyeres to the lowest point of the mouth. These dimensions for the two classes are: Class IV, large mouth, 7 ft. 11 in.; Class V, 8 ft. 9.25 in.

These we believe to be a few of the basic principles which have been demonstrated by the experimenting and practice at this plant.

We wish to acknowledge our indebtedness to Peter Thill, A. T. Elliott, and James Moore, who have been members of the staff from the very early days of the plant and are so still, for their recollections and descriptions of the early construction and practice.

DISCUSSION.

BRADLEY STOUGHTON, New York, N. Y.:—I would like to ask Mr. Krejci whether they have tried mixing coke dust with the lime in those converters and, if so, if any advantage has been found. It has been a very great advantage in the steel converters where coke dust has been mixed with the lime near the tuyeres. Of course the agent that causes the corrosion at that point is oxide of iron, and coke seems to prevent the formation of the oxide of iron and so protect the tuyeres. It would be interesting to know if it had the same effect in the copper converters.

MILO W. KREJCI:—We have never tried it here. At Anaconda the tuyere section was formerly put in by mixing magnesite with tar and ramming that into the tuyere section, but they have abandoned that. Might I ask how you would apply the coke dust? How get it into the tuyere section? How hold it there?

BRADLEY STOUGHTON:—The coke dust is mixed with silica and clay and then rammed in and burned. I suppose in your converters it would be mixed with the magnetite or magnesite and then agglomerated with the material in place.

MILO W. KREJCI:—We never tried it here.

PROF. JOSEPH W. RICHARDS, South Bethlehem, Pa.:—I would like to ask how thick this deposit of magnetite is on the lining. How thick a layer would be practical?

MILO W. KREJCI:—About 2 in. would be sufficient.

SOLOMON LE FEVRE, Mineville, N. Y.:—I was interested to hear of the use of magnetite in connection with the copper smelter. I am interested in the mining of magnetite, but was not aware that it had been used at all by the copper metallurgists.

Magnetite in lump form is used largely for bottoms of open-hearth steel furnaces. The lumps are filled in with a mortar of hematite ore. The chief advantage of the Mineville, N. Y., lump ore is that it glazes over when the heat strikes it, instead of decrepitating as some ores will. I see no reason why it should not be used for copper reverberatory furnaces.

Perhaps in the converters, if after the brick were heated the magnetite could be blown in with something like the " cement gun " it would immediately stick fast and form the glaze of magnetite sought for over the brick.

MILO W. KREJCI :—We might say to Mr. Le Fevre that we had discussed the advisability of trying to get some combination of this magnetite in the reverberatory, but we haven't seen our way clear to get it in there, to have it stick.

HERBERT HAAS, San Francisco, Cal. (communication to the Secretary*): An authoritative historical review of the development of a process is not only of great interest, but also of great value, as it preserves for future workers a record of what has previously been done. A detailed history of metallurgical processes, chronicling failures as well as successes, is particularly valuable, and Messrs. Wheeler and Krejci's paper on Great Falls Converter Practice, past and present, is a valuable contribution to the literature of the metallurgy of copper. The great value of authentic historical records of the development of important industries, chiefly the metallurgical and mechanical industries, is beginning to be realized in an increasing measure by engineers, and the Institute would confer great benefit upon the copper industry and the science of copper metallurgy if it would undertake, with the co-operation of our eminent copper metallurgists and specialists, the publication of a standard work on the history of copper, that would rank with L. Becks's *Geschichte des Eisens* (History of Iron), or Conrad Matschloss's *Die Entwicklung der Dampfmaschine* (The Development of the Steam Engine) and his *Beiträge zur Geschichte der Technik und Industrie* (Contributions to the History of Technology and Industry).[1]

Much of our information on copper is scattered through many publications, and a great portion thereof deals more with analytical

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[1] Four volumes of the latter have appeared to date ; Julius Springer, Berlin.

and chemical data than with the evolution in the mechanical design and construction of apparatus in use at different times. Papers that record and give due weight to the mechanical development in the metallurgy of copper are of particular value. The recent example set by Montana engineers in contributing so many excellent papers to the *Transactions* should be emulated by other workers in that field of metallurgy.

This discussion relates only to the latest development of Great Falls converter practice, and I shall confine the same at present to rate of oxidation, *i. e.*, speed of converting; number and size of tuyeres; effi-
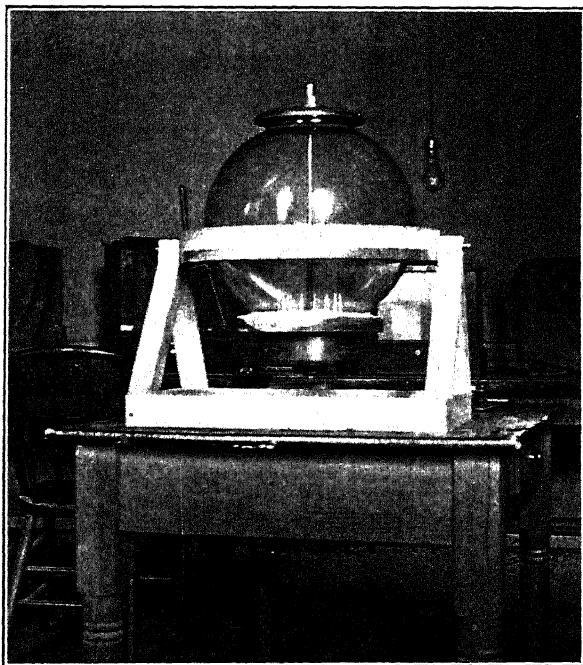
FIG. 19.—EXPERIMENTAL GLASS MODEL OF CONVERTER.

ciency of air; and air volume. In December, 1912, and January, 1913, and again in May, I conducted a series of experiments with a glass model of a converter, Figs. 19 and 20, to study the effect of size, direction, arrangement, and number of tuyeres on the circulation of the converter bath (in my experiments, water, brine, and mercury were used), and evolved a new converter construction. This work and the considerations in view were embodied in a paper, A Proposed New Form of Converter and Modification in Bessemerizing Copper Mattes. It was my intention to have this paper ready for the Butte meeting, but as I have not until the present date been able to com-

plete it, owing to pressure of other work, I trust that certain portions
thereof bearing upon this discussion may here be quoted.

Rate of Oxidation, or Speed of Converting.

This is primarily dependent on the speed with which oxygen (air)
is pumped into the converter.   During the first period, when blowing
on matte, we have the reaction :

$$x\ Cu_2S.FeS + 3\ O = x\ Cu_2S + FeO + SO_2$$
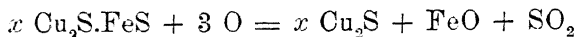
and during the second period, when blowing on "white metal":

$$Cu_2S + 2\ O = 2\ Cu + SO_2$$
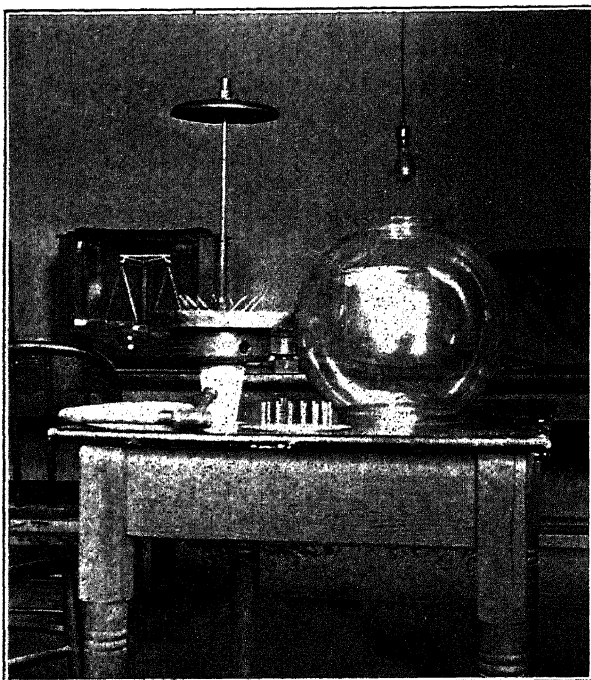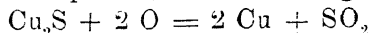
FIG. 20.—SEPARATE PARTS OF MODEL.

For every pound of oxygen forced into the converter there will be
oxidized, according to this formula, 1.17 lb. of Fe and 0.67 lb. of S,
with the ratio of $\dfrac{1.17\ Fe}{0.67\ S} = 1.75$, during the first stage of the process,
and one lb. of S during the second stage.   In actual practice the
ratio $\dfrac{Fe}{S}$ is more nearly 2; $i.\ e.$, twice as much iron as sulphur (by
weight) is oxidized.   This is due to the fact that nearly all mattes do
not conform to the ideal composition $x\ FeS.Cu_2S$, in which all the cop-
per and iron is combined with sulphur (as Cu_2S and FeS), but have

metallic copper dissolved in the FeS; the greater the amount of FeS, the greater is the amount of metallic copper that can be so held in solution.

As converting progresses, the amount of FeS diminishes, and with it the amount of metallic copper that is held in solution. That is, the relative amount of copper combined with sulphur increases and the amount of metallic copper decreases as the FeS diminishes, until practically all FeS is eliminated, and blowing on "white metal" begins. This is equivalent to a decrease in the rate of elimination of the sulphur, and an increase in the rate of elimination of the iron. A portion of iron already oxidized is also introduced with the siliceous flux, which is not properly chargeable to iron elimination by air oxidation. By computing the rate of oxidation on the basis of the formula FeS + 3 O, the converting speed would really be somewhat slower than in actual practice, but the above figures are sufficiently accurate to determine converter capacity.

Thus, by forcing 200 kg of oxygen per minute into the converter, 234 kg. of Fe per minute would be oxidized to 300 kg. of FeO (assuming that no Fe_3O_4 is formed), and 133 kg. of S per minute to 266 kg. of SO_2. Two hundred kg. of oxygen corresponds to 870 kg. of air, equal to 673 cu. m. (at standard conditions of temperature and pressure), or 24,000 cu. ft., of air per minute, which corresponds closely to the air volume used in the latest 20-ft. Great Falls converter. The proper distribution or diffusion of so large a volume of air through the converter charge is not only necessary to secure the required speed of reaction, but to blow on shallow charges with low air pressure, to prevent the throwing out of the converter charge and to avoid mechanical losses (shot copper and matte), accretions on and fouling of the converter mouth, and undue localization of oxidation, with increased wear on the lining in the tuyere zone. It is to this subject pertaining to copper-converter construction that I have given my attention since the introduction of the large Peirce-Smith basic-lined converters, and the one more recently put into operation at Great Falls.

My own experiments with the glass model converter confirmed what Messrs. Wheeler and Krejci described with reference to the circulation in their small experimental tank with glass bottom. That the air blast would not penetrate the charge appreciably in a forward (horizontal) direction before it rises is to be expected on purely physical grounds, as the mass ($m = \dfrac{W}{g}$) of air compared with that of water is very small, and still smaller as compared with that of matte, so that the force stored in the air is soon spent, owing to the resistance offered by the matte. The vertical component of the reacting

forces, due to the low specific gravity of the air, which has only about $\frac{1}{20000}$ of the weight of an equal volume of matte when heated to the temperature of the converter contents, is infinitely larger than the horizontal component, so that the resultant will be practically vertical; i. e., the air will rise almost immediately.[2]   It will rise more quickly through matte than through water, provided viscosity is not an impeding force.   Such an impedance is offered by the converter slag, which is viscous compared with slags made in furnace operations. How the converter slag may affect the speed of converting will be shown below.

" To set up a circulation in a direction opposite to that described as had been thought probable " by Messrs. Wheeler and Krejci, is hardly feasible, for other reasons than that already given above, which reasons will also explain why the circulation in a side-blown copper converter is precisely of the nature observed by Messrs. Wheeler and Krejci in their experiments.   In a converter this circulation is greatly intensified.

One of the considerations that led me to design a bottom-blown converter was the apparent localization of the oxidation in the present type of side-blown converter.   This is especially noticeable in the Great Falls converter, where the tuyeres are confined to a relatively small portion of the inside converter perimeter, and the tuyere orifices are from 8 to 15 ft. removed from the opposite side of the converter.   In the large Peirce-Smith converters this distance, as a rule, is 10 ft.   Here the localization of the oxidation is not quite so pronounced, as the tuyeres are spaced over the entire length of the cylinder (from 20 to 30 ft. inside) and a much smaller air volume is delivered.   Notwithstanding this condition of concentrating the blowing to one side of these converters, new matte is constantly brought into contact with the air.   Comparing the density of air with that of water, matte, "white metal," and blister copper, we find that at ordinary temperatures, say 15° C., 1 cu. m. of water is 800 times heavier than 1 cu. m. of air, that matte is 3,600 times heavier, " white metal " 4,400 times heavier, and blister copper 6,800 times heavier than an equal volume of air.   This air, when pressed into the converter, is heated to the temperature of the converter contents (from 1,150° to 1,200° C.), as the result of which the gases expand to about five times the original air volume.   Thus the gas becomes correspondingly lighter, the relative weights of equal volumes of matte and gas being about as 18,000 to 1.   This large volume of gas is

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[2] I called attention to this action of the air in a side-blown converter in a previous article, The Vortex Copper Converter, *Engineering and Mining Journal*, vol. lxxxix., No. 19, p. 972 (May 7, 1910).

principally confined to the tuyere zone, and the matte near the tuyere zone is rendered less dense thereby. The matte in the other part of the converter, which is not diffused by gas, retains its normal density, and exerts, therefore, a greater hydrostatic pressure. As this is equal in every direction, at any given height, a differential in pressure exists between this matte of normal density and the "wall" of gas-diffused matte of lower density. This condition will set up a flow in the direction of the tuyere-zone which will bring into contact with the air a constant fresh supply of matte. At the tuyere level, where the hydrostatic pressure is greater than at points higher up, this flow, or circulation, will ·be more rapid. The large volume of gas will carry with it and bring to the surface of the matte bath a certain amount of matte and create a violent wave motion in a direction away from the tuyere side. This localization of the oxidation and circulation at the tuyere zone causes increased wear on that portion of the
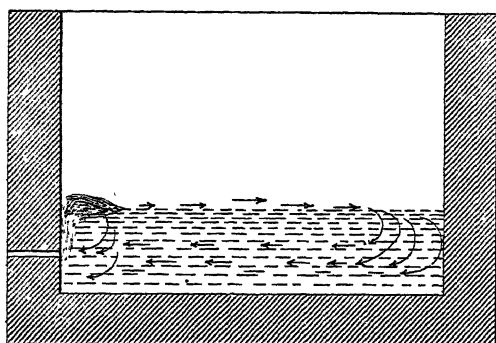
FIG. 21.—CIRCULATION OF THE BATH IN A SIDE-BLOWN CONVERTER.

converter lining. In Fig. 21 the circulatory action above described is illustrated.

Arguing that the rate of conversion could be increased, and at the same time the intense localization of the oxidation in so small an area of the large converters used at present greatly reduced, I designed a bottom-blown converter, the salient points of which are shown in Figs. 22, 23, and 24.

Referring to Fig. 22, it will be at once noticeable that the tuyeres are not distributed over the entire converter bottom, but are confined to a concentric area, in size about one-quarter of the total area of the bottom. This confines the oxidation and the ascending gases to a central column of the converter bath. The gases here have the same effect of materially decreasing the density of the matte as has been explained above for the side-blown converter, but with this marked difference: The outside "ring" of matte inclosing the central

" geyser," having a greater density and therefore greater hydrostatic pressure, is continually flowing toward the tuyere bottom, and ascends with a large volume of gas, overflowing at the surface and spreading toward the periphery.

In Fig. 23 this tuyere bottom is elevated, the brickwork being supported by a hollow, cast-steel truncated cone, which serves as a wind box. This elevated tuyere bottom forms an annular sump, in which
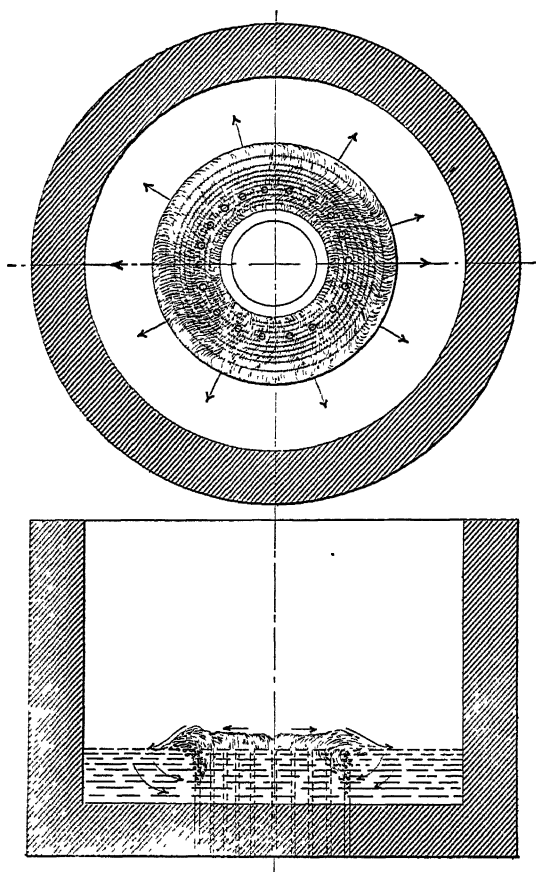
FIG. 22.—PLAN AND SECTION SHOWING CIRCULATION OF THE BATH IN
HAAS BOTTOM-BLOWN CONVERTER.

can be stored a considerable tonnage of matte. With this raised tuyere bottom, it is possible to blow on a shallow bath of matte and yet have a large volume of matte stored in the converter. This matte acts as a heat accumulator, and will be particularly useful when concentrates are to be smelted in the converter by blowing them into the matte through the tuyeres. This feature, to which my converter is especially applicable, I shall discuss at greater length in my paper.

Suffice it to say here, that with the wind box closed, and in the absence of tuyere valves and tuyere punching, there could be no objection to this practice on the ground of mechanical losses through the tuyere valves when the tuyeres are being punched.

My converter is also provided with a large slag mouth, a short distance above the slag line. Instead of tilting the converter completely to skim the slag, or blowing for too long a period on matte and letting the slag accumulate, this converter would be tilted at frequent intervals to pour off the slag. To accomplish this, the converter will have to be tilted only a few degrees. The removal of the slag at frequent intervals appears especially desirable, as it has been observed that the rate of oxidation falls off if the slag is left to accumulate and increases
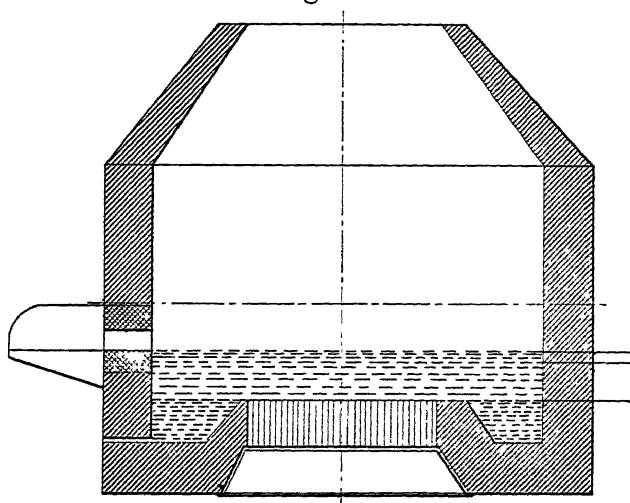
FIG. 23.—ELEVATED TUYERE BOTTOM FOR BOTTOM-BLOWN CONVERTER.

with the bath skimmed clean of slag.[3] This may be due to the fact that the SO_2 gas may be kept included in the matte, as it cannot pass off so freely through the viscous slag, and thus vitiates the oxidizing power of the air.

The static column is also kept low by a frequent removal of the slag. The converter would be worked very much as converters are worked at present. Frequent charges of matte are given, until there is a considerable accumulation of white metal. The peculiar feature, when working with the elevated tuyere block, is the slight increase in the height of the matte column above the tuyere block, as with the elimination of a portion of the sulphur and of the iron the matte shrinks in volume. When blowing on "white metal" begins, the

[3] Carr B. Neel: *Engineering and Mining Journal*, vol. xci., No. 14, p. 707 (Apr. 8, 1911).
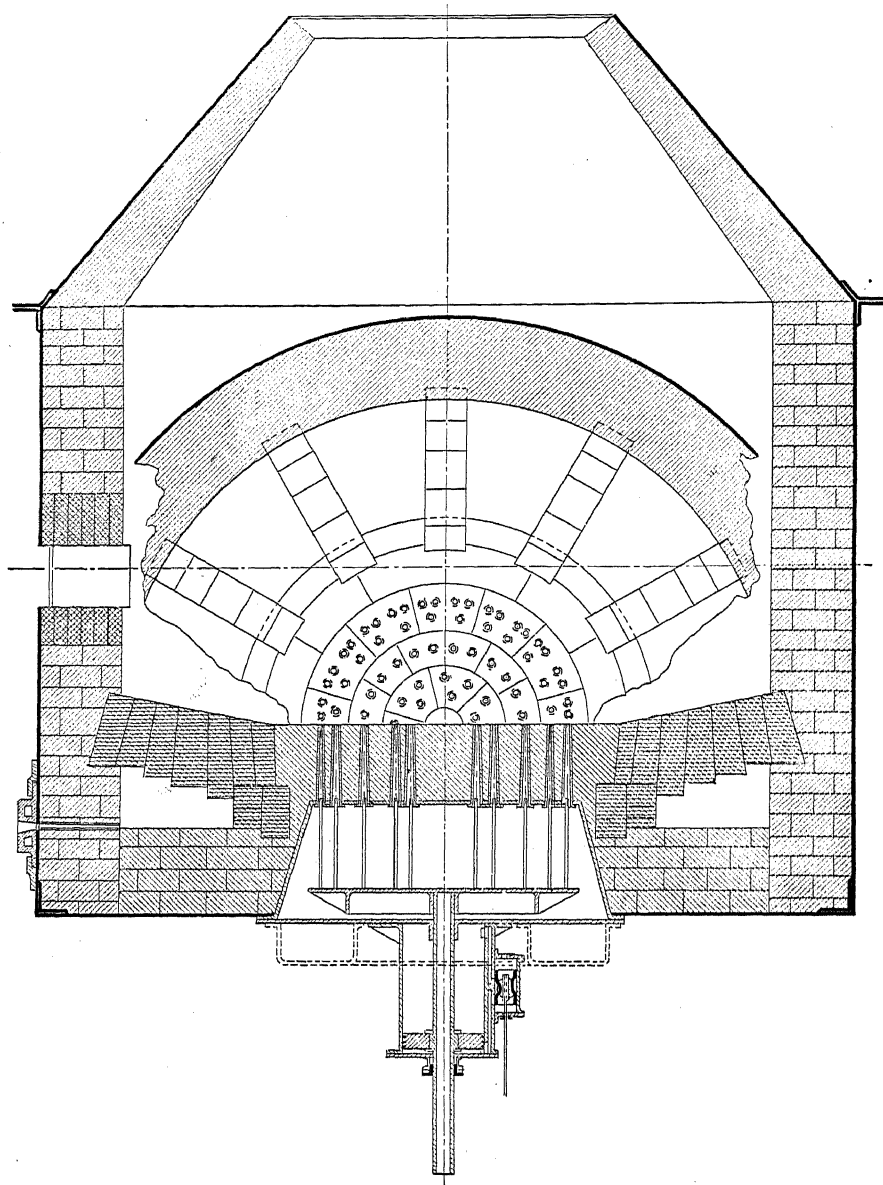
Height, 15 ft. ; helmet, 7 ft.   Diameter, shell, 20 ft. ; inside, 16 ft.   90 tuyeres,
1.5 in. in diameter.   Storage capacity of sump, approximately 264 cu. ft.,
or about 70 tons of copper.

FIG. 24. —HAAS CONTINUOUS CONVERTER WITH VERTICAL TUYERES AND
AUTOMATIC TUYERE PUNCHER.

height of the column above the tuyeres becomes constantly less, with a constant reduction in air pressure required, notwithstanding the fact that the density increases at the same time. This is due to a certain relationship between the area and volume capacity of the converter, and the height of the tuyere block above the converter bottom proper.

The method of mounting and operating the converter is not different from that now in use. The shell is connected with large riding rings resting on rollers, or riding wheels. To one of the riding rings is bolted a gear ring, which is engaged by a pinion that is operated by a further reduction of gears from a direct-current motor. The drive can also be made hydraulic. The converter has enough of a belly to hold the converter contents when turned down in the direction opposite to that of the slag mouth, so that it has all of the advantages of the converters now in use. To secure the best possible distribution of the air through the matte, and to attack as large a matte surface as possible, I have provided my bottom-blown converter with a large number of tuyeres of small cross-sectional area. This was also done to avoid the formation of " noses," which my investigations lead me to believe are wholly due to the large volume of air introduced per tuyere in the present type of side-blown converter. By establishing a proper ratio between the weight of air introduced through each tuyere per minute and the weight and size of a "column" of matte immediately above and surrounding each tuyere orifice, chilling should be prevented. The weight of air introduced per minute through each tuyere is reduced to a point at which enough heat can be transmitted from the matte to the air to heat the latter and still leave the matte surrounding the rising air currents at the tuyere orifice in a molten condition. The matte flowing toward the tuyere bottom would make up any deficiency in heat. In the side-blown converter this new matte comes from one side only, the other side being bounded by the converter wall. By subdividing the air currents the air surface exposed to the matte will be greatly increased, at the very point when it enters the matte, and the air will thus be more readily heated. It will also permit of working with a shallower charge, as there will be less concentration of air at a few points, with the possibility of a portion of this concentrated air volume escaping without all of its oxygen having been utilized, unless it is made to travel through a deeper matte bath to prolong its contact with the matte. I shall not use space at present to set forth thermo-chemical calculations incorporated in my paper, which confirm my view that a copper converter can be bottom blown without having to resort to any punching of tuyeres.

Messrs. Wheeler and Krejci are quite correct in saying that "with proper distribution of the air through the charge the speed of the reactions in the converter is undoubtedly beyond any practical rate of supplying the air." While it is only within recent years, beginning with the introduction of Peirce-Smith basic-lined converters, that the volume of air blown into copper converters per minute was materially increased, from 800 to 1,000 kg. of air per minute (22,000 to 27,400 cu. ft. per minute) has been the rate of air supply to 20- and 25-ton Thomas (steel) converters for a number of years. This is a size no longer uncommon in large steel works which use the Thomas process. These converters are by no means as large as the 20-ft. Great Falls converter, but they convert considerably faster, the elapsed time being greatly reduced. While the 20-ft. Great Falls converter has about three times greater air supply than is usual with the Peirce-Smith and the smaller 12-ft. size Great Falls converters now being installed in various works, I look for a greatly increased air supply to copper converters, as the use of magnesite linings will make this increased converting speed possible. So-called "needle bottoms," in contradistinction to "tuyere bottoms,"[4] have been commonly used for a number of years in Thomas converters with great success, and they are not only considered a great advance over the old style tuyere bottoms, but one of the principal recent advances in Thomas converter practice.[5] These needle bottoms have a large number of small tuyeres, from 250 to 300 of them in 20- to 25-ton converters, their diameter being from 15 to 18 mm. Thus the amount of air introduced per tuyere per minute varies from 3 to 3.5 kg., whereas in the Peirce-Smith converter it is from 6 to 12 kg., and in the latest Class V 20-ft. Great Falls converter about 14 kg., with 62 1.75-in. tuyeres, and about 34 kg. with the number of tuyeres reduced to 26 (2.25 in.). From the very start of copper converter operations the weight of air per minute per tuyere has been larger than in steel converting, and as long as an acid lining was used, that corroded rapidly, a limited number of tuyeres was desirable for various reasons. The formation of noses and clogging of tuyeres was pronounced, as corrosion or "slagging" of the lining was most intense in the tuyere zone, due to the circulation set up as described above. This made constant punch-

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[4] The tuyere bottoms usually have, not a few large tuyeres, but a number of tapered tuyere bricks, each containing from 10 to 20 small tuyeres, from 0.25 to 0.5 in. in diameter. If a brick is damaged, another one may be inserted in its place. This has the same effect as a few large tuyeres would have of concentrating the volume of air in a few places. • In the "needle bottoms" the tuyeres are evenly distributed over the entire bottom. See F. W. Harbord, *The Metallurgy of Steel*, 2d ed., pp. 9 *et seq.*, 60 *et seq.*

[5] 30 Jahre Thomas-Verfahren in Deutschland, *Stahl und Eisen*, vol. xxix, No. 38, p. 1486 *et seq.* (Sept. 22, 1909).

ing necessary, and for reasons of construction and attendance a limited number of tuyeres was ' desirable. In basic-lined converters. slagging does not occur in the tuyere zone, but above it, so that the air blows only on matte, and if the amount of air passing through each tuyere per minute be limited sufficiently I can see no reason why punching should be any more necessary than it is in steel converting.

To heat 1 kg. of air from 0° C. to 1,150° C. requires.
$$\frac{0.303 + 0.000027 \, t \times 1,150° \, C.}{1.293} = \text{(in round numbers)} \, 300 \, \text{kg-cal.}^6$$
It will therefore be seen that in the Great Falls converter, with 62 1.75-in. tuyeres, and an air supply of 14 kg. per minute per tuyere, 4,200 kg-cal. have to be supplied every minute by the matte for each tuyere (70 cal. per second), and with 26 2.5-in. tuyeres. this is increased to 10,200 cal. per minute per tuyere (170 cal. per second). This is undoubtedly far in excess of the rate at which the matte can transmit the heat before it will chill around the tuyere orifice, especially so in view of the close spacing of large tuyeres in side-blown converters and the concentration of air volume.

In the modern Thomas converter, due to the higher temperature, about 400 cal. are required to heat 1 kg. of air, so that here only from 1,200 to 1,400 kg-cal. per minute per tuyere have to be supplied. In my converter I propose to keep it between 800 and 1,000 kg-cal. per tuyere per minute.

Taking as an example my converter of 20 ft. outside diameter and 16 ft. inside of lining, in which I restrict the tuyeres to a circular area 8 ft. in diameter in the center of the converter, thus forming a tuyere bottom 50 sq. ft. in area, which is surrounded by the balance of the converter bottom, of some 150 sq. ft. in area, and using 300 18-mm. tuyeres, I would have a total tuyere area of some 120 sq. in., or
$$\frac{144 \times 50}{120} = 60 \text{ sq. in. of matte surface per 1 sq. in. of tuyere area.}$$
It is difficult to figure the exact relation between tuyere distribution and the matte surface belonging to it in a side-blown vertical converter, as all tuyeres are arranged in a line lying in the same plane through the vertical wall of the converter.

Allowing 240 in. as the length of the arc in which the tuyeres of

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6 As nitrogen forms the largest portion of the converter gases, and the mean specific heat of SO_2 is greater than that of air, the fact that part of the oxygen of the air combines with FeO may be neglected, as this way of figuring agrees within 5 per cent. with the amount of heat carried off by the waste gases. It is reasonable to suppose that the air is heated to the temperature of the matte before oxidation is rapid. In the above calculation, the loss in heat of the air on expanding when leaving the tuyere orifice was neglected.

the Great Falls converter are placed, and allowing the same distance
above and below each tuyere as the distance between centers of tuy-
eres, we would have an approximate surface of $8 \times 240 = \dfrac{1,920}{157} =$
12.2 sq. in. of matte surface per 1 sq. in. of tuyere area, and $\dfrac{1,920}{106} =$
18.2 sq. in. of matte surface per 1 sq. in. of tuyere area, with the 62
and 26 tuyeres respectively.

With this subdivision of the air currents, the air surface exposed to
the matte when the air leaves the tuyeres is also greatly increased.
In the Great Falls converter this is only 184 sq. in. for 1 in. length
of the projected air current (product of 1 in. length $\times$ periphery of
tuyere opening), whereas it is 670 sq. in. in my converter, or 3.64
times as large. This will influence very favorably the speed of re-
actions.

Large Tuyeres versus Small Tuyeres in Converters.

"Another point of superiority of a few large tuyeres, within the
limits of working, over many small ones, is in the decreased resist-
ance to the air, and consequently a lower necessary pressure."

I agree with the first part of this statement by Messrs. Wheeler and
Krejci, and if we were dealing simply with a case of selecting the
proper size of pipes to accommodate a certain flow of air large pipes
would be dictated on the ground of power economy. But it does
not follow that if a large number of small tuyeres were used in a con-
verter a higher working pressure would be necessary, which is im-
plied in the closing part of the above-quoted statement. With proper
modifications in design, on account of the altered conditions intro-
duced with the use of a large number of tuyeres, the *total* frictional
losses in my type of converter need not be greater than those in the
side-blown type, which uses a few large tuyeres.

The principal power loss, or drop in pressure, is at the tuyere ori-
fices, and not in the tuyere pipes themselves. This power loss is
chiefly due to the contraction of the tuyere orifices by frozen matte
adhering to them, which makes the orifices not only smaller in area,
but very rough, thereby greatly increasing the velocity and frictional
resistance. This increase in the velocity head and friction head
brings about a drop in pressure, or loss in effective power; that is,
the power used in compressing the air is mainly spent in imparting
this increased velocity to the air, causing a drop in pressure and a
decrease in the air volume delivered. After the air pressure at the
tuyere orifices has been lowered through loss of power represented

by the velocity head and friction head, there must still be sufficient excess pressure left to overcome the various resistances offered by the matte, which, in addition to the actual static head, are due to surface tension, viscosity of the liquid, etc. As I have already stated, I believe that it is the large volume of air flowing through each tuyere that is responsible for chilling and the consequent formation of noses; of this chilling and its effect in increasing the power requirements there is no doubt, as we have ample illustration of it in operating basic-lined copper converters, where the tuyeres must be punched frequently to keep them open and to prevent the air pressure from rising rapidly. As the charge of the basic-lined converter must be kept at a relatively low temperature (1,150° C.) to prevent excessive corrosion of lining, this tendency of the air blast to form noses is increased. As explained before, the character and form of an orifice through which air flows influence frictional losses tremendously; Weisbach's, and later Zeuner's, exhaustive investigations on the flow of water and gases (air, steam, etc.) through orifices have shown how much the character and shape of an orifice influence the contraction of the vein or flowing stream and friction, and determined a large number of empirical coefficients, to make allowance for varying conditions. In the case of the converter we are dealing with especially complex conditions, which are constantly changing, so that figuring the theoretical and effective velocities and weights of the air delivered through converter tuyeres by Zeuner's formula[7] is of little practical value. If it were a case of determining the amount of flow of air through a nozzle from an air receiver or air main, this could be done quite accurately.

That frictional losses in converting are quite large is evident from the fact that the ratio between static matte pressure and total air pressure used varies from 0.2 to 0.4; i. e., an air pressure from 5 to 2.5 times greater than the pressure due to static head is required.

Part of this excess pressure is, of course, requisite for safety to avoid filling of tuyeres. I figured the drop in pressure due to friction in the tuyere pipes of the Great Falls converter and bottom-blown converter of my type, using Lorenz's formula.[8]

$$\frac{\varDelta P}{Pm} = \alpha \frac{Lv^2}{dT}$$

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[7] For a full explanation of the derivation of this formula, which is a subject beyond the compass of this communication, consult the works of Dr. Gustav Zeuner : *Technische Thermodynamik*, vol. ii., pp. 137 to 186, and vol. i., pp. 212 to 262. A. von Ihering : *Die Gebläse*, 2d ed., Part II, pp. 708 to 744. *Hütte*, 19th ed., vol. i., p. 326 *et seq.*

[8] *Hütte*, 19th ed., vol. i., p. 335 *et seq.*

In which ΔP = drop in pressure.

Pm = mean pressure in pipe.

L = length of pipe in meters.

v = velocity of flow in meters per second.

d = diameter of pipe in millimeters.

T = absolute temperature = $273 + 0°$ C. [pipe.

a = empirical coefficient, depending on diameter of

20-ft. Great Falls Converter.	20-ft. Haas Converter.
26 2.25-in. Tuyeres.	300 18-mm. Tuyeres.
$d = 57$ mm.	$d = 18$ mm.
$L = 0.8$ m. (length of tuyere pipe).	$L = 0.8$ m.
$v = 170$ m-sec.	$v = 150$ m-sec.
$T = 273° + 100°$ C. $= 373$.	$T = 373$.
$a = 0.042$.	$a = 0.050$.

$$\frac{\Delta P}{Pm} = 0.042 \; \frac{170^2 \times 0.8}{57 \times 373}$$

$$\frac{\Delta P}{Pm} = 0.05 \; \frac{150^2 \times 0.8}{18 \times 373}$$

$$0.042 \; \frac{23,120}{21,261}$$

$$0.05 \; \frac{18,000}{6,714}$$

$$= 0,0458 = 4.58 \text{ per cent.}$$

$$= 0.1340 = 13.40 \text{ per cent.}$$

$\Delta P = Pm \times 0.0458 = 0.0275$ kg-cm.^{2} $\Delta P = Pm \times 0.134 = 0.0804$ kg-cm.^{2}

In both cases a working pressure of 0.6 kg-cm.^{2}, or 8.53 lb. per square inch, has been taken as a basis of comparison. (1 kg-cm.^{2} or metric atmosphere $= 14.2234$ lb. per square inch.)

The formula $h_w = f L \dfrac{v^2}{4d}$ gives somewhat different values. In this formula h_w is the "head" expressed in millimeters of water column required to overcome the resistance or friction offered by the pipe to the flow of air, f is an empirical coefficient of friction $= 0.003$, L = the length of the pipe in meters, v is the velocity of flow in meters per second, and d is the pipe diameter in meters.[9]

According to this formula, using the same values as above, the frictional head for the Great Falls converter (2.25-in. tuyeres) is 305 mm. of water, or 0.0305 kg-cm.^{2}, and for the Haas converter, with

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[9] See A. von Ihering: *Die Gebläse*, 2d ed., part II., p. 660 *et seq.* (Julius Springer, Berlin).

Knipping: *Zeitschrift des Vereines deutscher Ingenieure*, vol. lvii., No. 31, p. 1218 *et seq.* (Aug. 2, 1913).

D. W. Taylor, U. S. N.: *Transactions of the Society of Naval Architects and Marine Engineers*, vol xiii., p. 9 *et seq.* (1905).

18-mm. (0.708-in.) tuyeres, 750 mm. of water, or 0.075 kg-cm.^{2} With the same gauge pressure, viz., 0.6 kg-cm.^{2}, the percentage losses would be 5.1 and 12.5 per cent. respectively.

This formula is not strictly applicable for these conditions when as high a pressure as is customary in converting is used, but is used mainly to measure frictional resistance in heads of water in ventilating and fan systems. The Lorenz formula, based on very extensive investigations, is used to measure the drop in pressure due to friction in pipe lines conveying compressed air, and should give values sufficiently correct for our purpose.

In both of these calculations, the velocity was figured according to the formula $V = \dfrac{Q}{A}$, in which V is the velocity of flow of the air in meters per second, Q the volume of air in meters delivered per second, and A the discharge area in meters. The air volume delivered to both converters was figured at 25,000 cu. ft., or 710 cu. m., per minute, the area as 106 and 120 sq. in. respectively. In reality, this (theoretical) condition is not attained, as a smaller volume would be delivered through this area, due to contraction of the stream and friction due to rubbing and turbulence of the air. The quantity would therefore only be $Q = \mu\,A\,V$, in which μ is a coefficient taking these resistances into consideration.

If this volume is to be delivered, additional pressure is needed to make up for this loss in delivery. In the case of the large tuyeres, there is an additional factor introduced, due to the contraction of the tuyere orifices themselves; i. e., a partial closing of them by frozen matte. This reduces the effective tuyere area constantly, bringing about an increase in velocity. This increased velocity can only be imparted by an expenditure of power. An increase in the velocity head $\dfrac{v^2}{2g}$ means a decrease in pressure. If the blowing engine were to continue to work at the same pressure at which it worked before the orifices were contracted there would be a lessened air delivery. If the air delivery is to be kept constant, the engines can deliver it only at an increased pressure. With a further contraction of the tuyere area, the pressure limit of the engine may be reached, and, notwithstanding a high working pressure, a smaller volume of air be delivered.

As can be seen from the above formulæ, velocity influences frictional losses more than any other factor, since friction increases as the square of the velocity, and only directly as the length and inversely as the diameter of the pipe. Thus with a 25 per cent. contraction of the

tuyere area frictional losses in the tuyere pipes of the Great Falls converter would be 8.4 per cent., and with a 50 per cent. contraction, 19.2 per cent.   These figures, being based on theoretical velocities and deliveries, are too low, but are near enough to illustrate how an increase in velocity affects frictional losses.   Losses on account of the roughness and shape of the orifices are not included in the above figures.   These figures also show that frictional losses need not be prohibitive, notwithstanding the use of a large number of tuyeres, provided such tuyeres are kept open by a decreased air flow through each tuyere and a moderate air velocity is maintained.   As the life of the tuyere bottom in a copper converter will be longer than that in a steel converter, where the conditions are extremely severe with temperatures up to 1,600° C., the tuyeres would be provided, as is done now, with smooth pipes, to keep frictional losses down.

In the foregoing I have discussed only the influence of pipe surface on the resistance to the flow of air.   But in converting we should also discuss the influence of the size of tuyeres on the probability of the liquid matte " breaking through " the pneumatic column, or " piston," in the tuyere pipes and filling them.   It may be stated axiomatically that the larger the diameter of the supporting air column, the greater is the danger of the matte flowing into the tuyeres, so that to prevent such filling of the tuyeres a higher air pressure is necessary with large tuyeres than with smaller ones.   This very tendency of the matte to flood large tuyeres will cause the operator to carry a safe margin of excess pressure to prevent this possibility.

With large tuyeres there exists always the danger of the charge being thrown out of the converter.   This was amply demonstrated in steel converter practice before small tuyeres evenly divided over the bottom were used.   Messrs. Wheeler and Krejci testify to that condition in the Great Falls converter in different portions of their paper. Even without the bodily throwing out of the charge, the converter mouth is more easily fouled and the mechanical losses are greater, as the amount of shot matte thrown against the top of the converter and out of it is greater than it would be in a converter with shallow bath and into which the air is introduced in small individual jets over a large surface.   The action is not so violent, as there is no concentration of a large volume of gas to a restricted area.

An increased blast pressure is also required in the vertical side-blown converter with a change in its position from the vertical.   If it is tilted up, the tuyeres furthest away from the axis around which the converter is tilted will blow on a shallower matte column than those nearer that axis, and if turned down, the reverse will be the case.

With the different resistances offered to the air at each tuyere orifice, the air flow will be concentrated through the tuyeres with least resistance, and the lack of a sufficient air flow through the lower and lowest tuyeres will cause these to be filled with matte, unless the blast pressure is increased. This condition has always demanded a higher air pressure for the vertical side-blown converter than for the cylindrical, in which latter all the tuyere orifices lie in a plane through the axis around which the cylinder revolves; no matter in what position it is, the hydrostatic pressure (though this may vary in magnitude) is equal at every tuyere. It is, of course, well known that it was the lower blast pressure required by the cylindrical acid-lined copper converter that led to its wide adoption, after the Copper Queen Consolidated Mining Co. installed the "Leghorn" type of shells at Bisbee, although the first Manhès-David converters installed at the Parrot works, and generally referred to as the "Parrot" converters, marked the introduction of the Bessemer process applied to mattes in America. The vertical shell is being adopted of late, because it lends itself better structurally than the cylindrical shell to securing the magnesite lining, and to absorbing the internal strains. These features, in view of the high first cost of the magnesite lining, outweigh the disadvantages due to any increase in air pressure required.

In the bottom-blown type I propose, the converter would remain in a vertical position, blowing on a uniform height of matte bath. In Thomas converters the "needle bottom" comprises the entire converter bottom. This obliges the operator to carry from 600 to 700 mm. of pig iron charge to blow on a large charge (from 20 to 25 tons). This represents a static pressure of from $0.06 \times 7 = 0.42$ kg-cm.^{2} to $0.07 \times 7 = 0.49$ kg-cm.^{2} (6 to 7 lb. per square inch) due to the liquid iron alone, figuring the weight of molten pig iron at 7, and requires a blast pressure of from 20 to 30 lb. per square inch.

In my design this needle bottom is confined only to a portion of a converter bottom of large diameter. I would use the same size, or even a larger size, needle bottom than is customary now for the Thomas converters, but by having a total bottom area at least four times larger I can hold the same tonnage with a static column only one-quarter as high, or a larger tonnage with only a slight increase in the height of the liquid. With the tuyere block elevated, although this is not at all essential, the matte volume that can be stored is increased, and yet a shallow matte bath maintained above the tuyere orifices. This makes it possible to work with low air pressures. Due to the circulation set up, the speed of converting will be accelerated. It is for this reason that I advocate the widest possible distribution of

the air, so that it can be used efficiently on a shallow charge. For this work, turbo-blowers direct connected to electric motors or steam turbines are particularly well adapted, as they admit of wide regulation with variations in the pressure as well as air volume, for which reasons they have found extensive application in European Thomas steel works as well as for blast furnaces. For the latter, on account of the higher fuel economy of the gas engines burning blast-furnace gas, the gas-engine-driven blowing engine is still preferred, whereas for converter service the turbo-blower is preferred on account of its better and wider range of regulation. This practice is beginning to be more extensively adopted in America. I understand that turbo-blowers are used at the Great Falls Reduction Works for converter service. The use of 30,000 cu. ft. of air per minute for a converter of the size here discussed should not be considered an unusual performance in view of present-day Thomas converter practice, provided the air is distributed in the manner indicated.

Tuyere Area.

While a large aggregate tuyere area is desirable, there must be a fixed relationship between the largest area that is permissible and the air supply to the converter per minute. If this area is exceeded, then the air supply will be insufficient to keep up the required pressure; *i. e.*, it is used faster than it is supplied and the tuyeres will be filled with matte. This condition is very much that of an air receiver from which air is drawn in greater amount than it is supplied by the compressor.

The following formulæ are commonly used to calculate the weight of air delivered through iron blast furnace and converter tuyeres, and the tuyere diameter, and are also applicable to the copper converter: [10]

$$W = \frac{13.6 \times 60}{10,000} \sqrt{\frac{2g}{R}} \; \mu \, F \sqrt{\frac{(b + h_2)\,(h_1 - h_2)}{273 + t}}$$

$$= 0.06664 \; \mu \, F \sqrt{\frac{(b + h_2)\,(h_1 - h_2)}{273 + t}}$$

In which $W =$ the weight of air in kilograms per minute delivered through the tuyeres.

$b =$ the barometric pressure in millimeters of mercury.

$h_1 =$ manometer (gauge) pressure in millimeters of mercury in the wind box at the tuyeres.

[10] Hütte, 19th ed., vol. ii., pp. 704 *et seq.*, 712.

$h_2 =$ the opposing pressure inside of the converter in millimeters of mercury; *i. e.*, hydrostatic pressure due to matte column expressed in millimeters of mercury.

$t =$ the temperature of the air in degrees C. in the tuyeres (this can be taken at 100° C., so that $T = 373$).

$F =$ the area of all tuyeres in square centimeters.

$g =$ the acceleration due to gravity $= 9.81$ m-sec.^{2}

$\mu =$ the coefficient of efflux. For tuyeres that are lined with smooth pipes, this can be taken at 0.8; for tuyeres in the lining, 0.75 should be taken.

R   is Regnault's constant $= 29.4$ m-kg.

For matte with weight of 4.5, 1 mm. matte column

$$= \frac{4.5}{13.6} = 0.331 \text{ mm. Hg.}$$

For " white metal" with weight of 5.5, 1 mm. "w. m." column

$$= \frac{5.5}{13.6} = 0.4044 \text{ mm. Hg.}$$

i. e., multiplying matte or "white metal" columns in millimeters by 0.331 and 0.4044 respectively, will give the corresponding mercury column in millimeters.

To figure the tuyere diameter necessary for a given working pressure, the following formula is used :

$$\frac{\pi}{4} d^2 = \frac{F}{n} = \frac{W}{0.06664 \, n \, \mu} \sqrt{\frac{273 + t}{(b + h_2)(h_1 - h_2)}}$$

in which d is the tuyere orifice diameter, and n the number of tuyeres.

Air Efficiency.

In addition to the " chemical " air efficiency, *i. e.*, the ratio between oxygen that actually combines with the impurities burned off and the amount of oxygen that is pressed into the converter, it would be interesting to know what may be termed the " volumetric " air efficiency of the Great Falls converter, *i. e.*, the ratio between the theoretical amount required to oxidize the iron, sulphur, and other impurities in the matte converted, and the amount of air actually supplied to the converter measured by engine displacement. While the air that is actually pressed into the converter is efficiently used, there are leakage losses between the engine and the tuyeres, chiefly in punching tuyeres, as the volumetric efficiency of the blowing engine working

at such low pressure should not be less than 97 per cent. While the air requirements vary with the composition of mattes, 100,000 cu. ft. of air (at standard conditions), or 850 kg. of oxygen, per ton of copper, will meet the requirements of most mattes treated at American copper-smelting works. Compared with this we have air consumptions ranging from 126,000 to 260,000 cu. ft. of air per ton of copper, or efficiencies varying between 80 and 38.5 per cent., both with acid- and basic-lined converters. This means that from 1.25 to 2.6 times the theoretically required air weight is supplied to copper converters. The first is excellent practice, the latter is poor work. At the Garfield plant of the American Smelters Securities Co., using Peirce-Smith converters, this air efficiency is reported to be around 77 to 80 per cent., which is very efficient practice.

With punching done away with in copper converting, a large source of loss of air will be eliminated, as leaks can be more effectively prevented with a tightly closed wind box without tuyere valves. Leaks would be mainly restricted to the universal air connection, which it is little trouble to keep tight when using stationary converter shells. The use of large converters minimizes air losses greatly.

Mechanical and Automatic Punching of Tuyeres.

I have previously called attention to the introduction of concentrates through the tuyeres. As these contain a varying amount of silica, slag would be formed at the very tuyere orifices, and under these conditions punching and the use of larger tuyeres would very likely become necessary. To do this punching mechanically and automatically, I have designed a tuyere-punching machine. Fig. 24 shows its salient features.

It consists of a plate to which are secured vertical tuyere-punching bars extending up into the conically enlarged tuyere pipes. The disk, or plate, with the punching bars is actuated by piston and piston rod working in an air cylinder supplied with high-pressure air (90 to 100 lb. per square inch).

All tuyeres are punched simultaneously by the forward (upward) movement of the piston and disks, the upward motion being reversed at the end of the piston stroke, when the punching bars are drawn back into a position of temporary rest.

The piston speed and the piston's upward and reversing (downward) motions are automatically controlled by a slide valve, operated mechanically from the tail rod. The time interval between punchings of tuyeres is controlled by an adjustable timing device, so that the

tuyeres can be punched at regular intervals, say every 5, 10, or 15 min., or whatever time interval is found to give best results.

As the plate with punching bars is located inside of the wind box, all tuyere valves and their leakages will be eliminated. The mechanical punching will also save the labor now used in punching tuyeres.

The air used for operating the piston, instead of exhausting into the atmosphere, preferably exhausts into the wind box at 21 to 27 lb. per square inch absolute, this terminal pressure depending on the working pressure of the air used in converting. An air receiver would have to be interposed between the converter and engine, which would act as a buffer when the tuyere pipes are momentarily partly closed during punching.

To exchange the punching rods quickly and to open the wind box and tuyere puncher to easy and quick inspection, the cast-iron plate, with its air cylinder attached, would be hinged. To let it down gradually, its travel would be controlled by a small air cylinder fastened to the converter shell, the air in which on being compressed acts as an elastic brake. To lift the plate back on to the hinged bolts the same cylinder would be used, its piston being forced back by compressed air.

The tuyere pipes would have bushings of chilled iron or manganese steel to prevent too rapid wear.

If concentrates are to be blown into the converter and the converting speed is not to be seriously retarded the air requirements have to be greatly increased. This is evident from figuring the oxygen requirements of the concentrate itself. A concentrate with Cu, 13; SiO_2, 25; Fe, 26; Al_2O_3, 5; and S, 30 per cent., would require, theoretically, 37.5 kg. of oxygen for every 100 kg. of concentrate, or 163 kg. of air, which is about 130 cu. m., or 4,600 cu. ft. If we were to treat 100 kg. of concentrate per minute, at least this amount of air would have to be supplied in addition to what is needed for converting the matte.

When treating a low-grade matte, very siliceous copper concentrates high in copper are very desirable, as these will displace siliceous ore usually fed through the converter mouth, and at the same time enrich the matte. Such a concentrate is that produced by the Miami Copper Co., and which was treated in this manner by the Cananea Consolidated Copper Co.

In view of the great future this practice has, large converters that can utilize efficiently and at moderate air pressure very large volumes of air will undoubtedly find increasing application in copper-smelting works.

Tuyere Bottom.

In Fig. 24 this bottom is made up of segments and held in place by a series of inverted arches. These arches will be tightened by the hydrostatic pressure of the matte, "white metal," or blister-copper; this pressure will correspond to that exerted by a column of liquid of the same height as the height of these arches. This pressure is about 4 lb. per square inch for matte and a total of about 1,700 lb. per arch. It may increase to nearly 7 lb. per square inch and 3,000 lb. per arch with blister copper. A better method of constructing the tuyere bottoms is that used by Thomas steel works, and as they have had 34 years' experience in the use of basic linings, their practice can be followed with profit. These bottoms are preferably made as a monolith and pressed into place with a hydraulic ram. A number of these are kept on hand, and the converter is so designed that they can be inserted and removed through the wind box. The distribution of so large a volume of air through this bottom, coupled with the low temperature at which a copper converter is blown, should not be without beneficial influence on the life of the bottom, as it should be cooled sufficiently to prevent too rapid erosion. After all, it is the magnesite consumption and cost per ton of copper that counts, and if by increasing the speed of converting the life of the lining should be decreased, consumption per ton of copper does not necessarily have to increase. In Thomas converters they treat tonnages with a converting speed that has as yet not been equaled in any copper-converting plant, and while their magnesite bottoms go faster, the consumption of magnesite per ton of steel is not greater than that per ton of copper in copper converters. It should be borne in mind that the work is very much more severe, with temperatures of 1,600° C. toward the end of the blow. This matter of constructing the tuyere bottoms I shall discuss more fully in my paper referred to above.

The building of so large a converter and the great increase in converting speed made possible thereby is undoubtedly one of the most important recent advances in copper metallurgy, and a great deal of credit is due Messrs. Wheeler and Krejci for their achievement.

With this large converter making possible the use of large air volume at low pressure they are nearer the realization of John Holloway's ambition to smelt pyrite ores in the "converter" without the use of fuel save that furnished by the pyrite.

REDICK R. MOORE, Mexico City, Mexico (communication to the Secretary *):—Messrs. Wheeler and Krejci have given an extremely

valuable history of the development of converter practice at Great Falls, Mont., and a *résumé* of the experiments which have led up to their present practice, which may be characterized as large upright converter vessels with basic lining and large tuyere area—the latter obtained by greatly increasing the diameter of the tuyeres as well as their number. They, however, do not give any data as to matte and slag analyses, blast pressure, utilization of blast volume (piston displacement), interior area of the tuyeres after protracted use, nor frequency of punching and size of punch bars used.

It seems to me that the addition of these figures would complete and enhance the value of their paper.

There must be sufficient blast pressure to overcome the static head of the column of matte and slag in the converter vessel and the friction head in tuyeres and pipes, together with an excess as a safety factor.

Excessive blast pressure results in a loss of both power and air efficiency (the latter on account of excessive leakages), and also, with tamped linings, in irregular wear.

At Aguascalientes, in 1904, experiments were made to determine the effect of enlarging the tuyere area, with a view to the reduction of power costs, and to determine the height of matte column above the level of the tuyeres necessary to secure the utilization of the oxygen of the blast.

The results of these experiments were presented to the Institute in a paper written in 1907, but not published. Briefly, the enlargement of the tuyeres to 1.25 in. in diameter, from about 0.75 in., resulted in a reduction of the blast pressure from about 18 lb. to about 12 lb., and an increase in production of about 20 per cent., maintaining the same engine speed. At the same time the life of the acid lining was increased greatly on account of more even wear and the absence of float silica.

Tuyeres 1.5 in. in diameter proved unsuccessful, as they weakened the lining too much and the end ones filled up in turning up to blow.

To secure the benefits of the enlarged tuyeres it was necessary to punch them frequently with a bar nearly the size of their diameter. The bar used was 0.75 in. in diameter with a head about 1⅛ in. in diameter.

The Aguascalientes converters were 8 ft. in diameter by 16 ft. upright, acid lined, and when newly lined were 3 ft. in diameter at the tuyeres.

The utilization of the oxygen of the blast (engine displacement)

varied from 67 to 83 per cent., with an average of six tests of about 75 per cent.

The delivery of air by the engine, as calculated from "cards" of the air end by the Master Mechanic, was 93 per cent. of the displacement.

It was calculated from the results of measurements and weights that with 6 in. of matte above the tuyeres the utilization of the oxygen of the blast was practically complete in these converters with an engine displacement of 5,600 cu. ft. per minute; barometer, 23.6 in.; temperature, 33° C.

The reduction of blast pressure was not in proportion to the increase in nominal tuyere area, even when punching with $1\frac{1}{8}$-in. headed rods, because of the formation of crusts in the tuyeres near their mouths and the formation of "noses," but the results were very gratifying.

The analyses given by Wheeler and Krejci (pp. 530 to 532) do not seem to conform to the theoretical. The table shows the ordinary converter reactions, together with the gases resulting if the oxygen of the blast is completely utilized for combustion.

The predominating reactions in the first or slagging period of converting copper-mattes are II and III of the accompanying Table I, while the subsidiary reactions I, IV, V, VI, VII, X, XIII, and XV to XXI, are going on simultaneously.

As the formation of Fe_3O_4, as shown by the products, is in excess of the reaction IV, the average gases for the period should contain more than 13.04 per cent. of SO_2 by volume and less than 13.90 per cent. In the second period of converting (white metal to copper) the predominating reactions are VIII and IX, while the subsidiary reactions XI, XII, XIV, XVI, and XVIII are going on simultaneously. As the formation of Cu_2O, as shown by the products, is in excess of the reaction XI, the average gases for the period should contain less than 20.8 per cent. of SO_2 and more than 19 per cent.

Throughout the operation the gases should contain free volatilized sulphur, PbS, ZnS, and AsS, as shown by the equations I, XVI, XVII, XVIII, and XIX.

With volumes of from 5,000 to 12,000 cu. ft. of air per minute (engine displacement), I have determined qualitatively the presence of these compounds in the converter gases. This is evidence that there was a complete utilization of the oxygen of the blast.

I have previously explained my reasons for believing that there is no SO_3 in converter gases.[11]

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[11] *Engineering and Mining Journal*, vol. xc., No. 6, p. 264 (Aug. 6, 1910).

A great many of the analyses given show wide differences from the theoretical and we are forced to the conclusion that either the theory is wrong or the analyses are wrong.

Not having any slag analyses, it is impossible to calculate any value for the CO_2 in the gases from the decomposition of carbonates, but from the slag analyses of a previous period, showing only 0.7 per cent. of CaO in the slag, it would seem that the CO_2 in the gases would be a negligible quantity.

The CO_2 from the atmosphere would be undeterminable by the ordinary methods of gas analysis.

It would seem to be very desirable to rigorously verify the analyses or determine the sources of error.

Table I.—Converter Reactions.

		Volatilized.	Gas.	
			SO2.	N.
			Per Cent	Per Cent.
I. $5\ FeS + Heat$	$= Fe_5S_4 + S$	S.		
II. $Fe_5S_4 + Cu_2S + O_{13}$	$= 5\ FeO + Cu_2S + 4\ SO_2$		13.90	86.1
III. $Fe_5S_4 + O_{14}$	$= 2\ FeO + Fe_3O_4 + 4\ SO_2$		13.04	86.9
IV. $2\ Fe_3O_4 + Cu_2S$	$= 6\ FeO + Cu_2 + SO_2$		100.00	
V. $Fe_3O_4 + Fe_5S_4$	$= 4\ FeO + 4\ FeS$	S.		
VI. $Fe_5S_4 + Cu_2 + O_{11}$	$= 5\ FeO + Cu_2S + 3\ SO_2$		12.52	88.4
VII. $5\ FeS + Cu_2$	$= Fe_5S_4 + Cu_2S$			
VIII. $Cu_2S + O_2$	$= Cu_2 + SO_2$		20.8	79.2
IX. $4\ Cu_2S + O_9$	$= Cu_6 + Cu_2O + 4\ SO_2$		18.52	81.4
X. $Cu_2O + Fe_5S_4 + O_{12}$	$= Cu_2 + 5\ FeO + 4\ SO_2$		14.90	85.1
XI. $2\ Cu_2O + Cu_2S$	$= Cu_6 + SO_2$		100.00	
XII. $Cu_2 + O$	$= Cu_2O$			100.0
XIII. $Fe + O$	$= FeO$			100.0
XIV. $Cu_2S + H_2O$	$= Cu_2O + H_2S$			
XV. $FeS + H_2O$	$= FeO + H_2S$			
XVI. $2\ H_2S + SO_2$	$= 2\ H_2O + S_3$	S.		
XVII. $3\ ZnS + O_6$	$= 2\ ZnO + ZnS + 2\ SO_2$	ZnS	14.90	85.1
XVIII. $2\ PbS + O_3$	$= PbO + PbS + SO_2$	PbS		
XIX. $3\ Fe_4As + 10\ FeS + O_{43}$	$= 22\ FeO + As_2O_3 + 9\ SO_2 + AsS$	AsS	9.80	90.2
XX. $CaCO_3 + Heat$	$= CaO + CO_2$			
XXI. $2\ MO + SiO_2$	$= (MO)_2\ SiO_2$ slag			
XXII. $S + O_2$	$= SO_2$		20.80	79.2

Monolithic Magnetite Linings for Basic Copper Converters.*

BY ARCHER E. WHEELER AND MILO W. KREJCI, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

Introduction.

THERE are two general methods in use for the production of metallic copper from matte which are worthy of consideration: (1) the Welsh blister process, and (2) the converter process.[1]

As practically all of the copper produced to-day is made in converters, we will give a brief outline of that process before proceeding with the special subject of this paper.

Matte may be roughly defined as a mixture of the sulphides of copper, iron, nickel, silver, etc.

In the converter process this is blown up to metallic copper in a modified type of Bessemer converter.

The conversion of matte consists mainly in, (1) the removal of sulphur in the form of SO_2, and (2) the slagging of the iron.

With this brief statement we come to the two subdivisions of the process of converting mattes:

Converting with acid linings.

Converting with basic linings.

Converting with Acid Linings.

This is carried out by lining the converter with siliceous rock or ore, which performs two functions: It protects the iron shell of the converter from the molten charge, and it furnishes the silica for combining with the iron as it is oxidized from the matte, to form a ferrous silicate. It follows from the second of these functions that this siliceous lining must be replaced from time to time, and in practice it has been necessary to reline converters every 8 to 40 hr., this time varying in different plants and even with different linings in the same plant, the life of a lining depending upon the grade of the matte treated, and upon the composition, the thickness, the tamping, and the drying of the lining.

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* U. S. Patent 1,068,470, process of forming furnace linings, issued July 29, 1913, to Archer E. Wheeler and Milo W. Krejci.

[1] Peters: *Practice of Copper Smelting*, p. 460 (1911).

The process was modified by the introduction of siliceous ore into the converter through the mouth, during its campaign, thus prolonging the life of the lining by furnishing some of the silica necessary to flux the iron. It never became possible in practice to prolong the life of the lining more than a few hours, due to the mechanical wear and wash of the heavy and hot charge.

It will thus be seen that the acid lining was expensive and cumbersome.

Converting with Basic Linings.

A few years ago basic linings of magnesite brick came into use. After the first demonstrations had shown the advantages their use spread rapidly, until to-day practically all of the converting of copper mattes is in basic-lined converters.

The principle of the basic lining is very simple and clear: namely, the use of a chemically basic material, in order to avoid chemical action between the lining on the one hand, and the charge and the products of the process, which are also basic, on the other. The necessary acid (silica) to carry on the chemical reactions is supplied by introducing siliceous ores, or other materials, into the converter, either through the mouth or through the tuyeres.

Although magnesite and other materials had been tried experimentally at the Boston and Montana Reduction Works at Great Falls, basic lining was not introduced permanently until Mar. 9, 1911, after it had been in use for a time at Baltimore, Garfield, and Anaconda.

The usual practice to-day is to use a magnesite-brick lining in the shell, varying from 9 to 30 in. in thickness in different parts of the converter, the thickest lining being at the tuyeres, on account of the intense chemical action and the high temperature at that point.

Unless the temperature is allowed to become too high there should, from a chemical point of view, be no corrosion of the lining. However, the converting process does not maintain a uniform temperature in the converter, the heat evolved from the chemical reactions being different at different stages of the process; and further, the process is an intermittent one, a charge being introduced, finished, and poured out of the converter, and then a fresh charge introduced, the result being varying temperatures, the extreme low temperature occurring in the interval between pouring one charge and introducing the next. These variations of temperature cause the brick to crack and spawl and break off, tending to thin the lining in places.

Again, if the temperature is allowed to become too high the brick become soft, and then mechanical wash and abrasion wear off the lining. This latter wear takes place under normal working condi-

tions, but not to the same extent as when the temperature is too high. Thus the theoretically indestructible lining is not practically so.

In order to guard against this wear and protect the brick it was the practice to heat the newly lined converter to a bright red heat with wood, coal, oil, or other fuel. When it had been brought to the proper temperature a charge of matte was blown to white metal in another converter and this white metal charged to the new converter, and then blown to copper, thus coating the lining with a thin layer of white metal and copper. This operation was repeated several times before the converter was considered ready for the regular service of blowing matte. This did not give very satisfactory results, as the melting point of the white metal and copper was low and any slight increase over the normal working temperature would result in melting off this coating, exposing the brick, which would then wear away. That this is not a satisfactory method is attested by the fact that there have been many basic linings which have lasted only a few days, whereas basic linings properly protected, as described later, have lasted over two years and are still in service.

We very soon deviated from this method of preparing a new converter to the practice of immediately blowing matte, blowing the charge for a few minutes until the temperature approached a dangerous point, then allowing the charge to stand for 15 or 20 min., then repeating the treatment, and continuing this cycle of operations until the brick had become coated with a layer of matte. This was just as good as the white metal treatment, and better from the point of view of doing away with the transfer of white metal.

Monolithic Magnetite Lining or Coating.

This method of protecting the brick lining of a converter is based on the fact that magnetite[2] is formed in artificial magmas, especially where the proportion of silica is low. Any excess of iron over that needed to combine with silica is likely to be deposited in the form of magnetite.

The melting point of magnetite is given by various authorities as from 1,527° to 1,538° C. The ordinary working temperature in a converter will rarely go over 1,200° C. unless improperly operated. It will be evident that, if we have a basic coating or covering over the brick lining, which coating has a melting point greater than that of the charge, and which can be put on at will during the process of

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[2] Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 327 (1911).

the blow, once a converter is lined, it should never be relined. This would be the case if 100 per cent. efficiency were possible.

The method of applying this lining to converters is as follows: The newly lined shell is heated up gradually to a bright red heat and charged with liquid matte, preferably of a low grade, say 35 per cent. The charge of matte is blown for 10 or 15 min., when the addition of cooling material will be found necessary, on account of the increase in temperature in the converter. The cooling is performed by the addition of cold matte. Alternate blowing and the addition of cold matte is continued until the charge is blown to white metal. No ore or other siliceous material has been added thus far in the blowing of the initial charge of matte, and therefore, with the lack of silica, the conditions have been right for the formation of magnetite.

It will now be observed that the interior of the converter has become coated with the magnetite formed during the blow. A fresh charge of liquid matte is added, together with a little less than the requisite amount of silica, and blowing is continued in the regular way until the brick lining is thoroughly coated and no brick is exposed. After this the actual silica required to flux the iron is used.

Under these conditions the lining should last a very long time, as the coating can be maintained by proper regulation of the converting temperature and the siliceous charge.

Should the coating become too thick, the silica of the charge can be increased, as well as the temperature, and a portion of the coating removed; and vice versa, in case the coating wears off and the brick becomes exposed.

In the blowing of the matte without ore, the iron becomes oxidized to Fe_3O_4 and the sulphur to SO_2. The working temperature of the converter can be kept down to normal by adding cold matte, converter cleanings, scrap, or ore, the material used depending on the operating conditions of the charge.

A detailed record of the basic converter practice at Great Falls up to June 1, 1913, is given in a previous paper.[3] The following table, containing parts of Table IV of the paper referred to, brings this information up to date.

[3] Wheeler and Krejci: Great Falls Converter Practice, this volume, p. 486.

Record of Basic-Lined Converters.

Converter.		Campaign.		Elap-sed Days.	Tons Cu.	Charges.	Tons Cu per Charge.	Total Time Operating.	
Class.	Bottom.	From.	To.					Hr.	Min.
IV.	B	Mar. 9, 1911	June 3, 1912	451	12,216.1	1,026	11.9	7,509	45
IV.	C	Apr. 29, 1911	Jan. 4, 1912	251	7,479.1	580	12.9	4 149	40
IV.	D	May 7, 1911	Oct. 9, 1913	886	17,246.3	1,314	13.1	9,756	35
IV.	A	June 22, 1911	Oct. 15, 1913	846	20,331.4	1,448	14.0	9,986	15
c III.	3	Nov. 16, 1911	Dec. 25, 1912	406	8,990.7	640	14.0	4,523	50
IV.	C	Mar. 17, 1912	Apr. 13, 1913	392	5,643	445	12.7	2,960	10
V.	A	Aug. 3, 1912	Dec. 17, 1912	137	2,835	88½	32.4	656	45
b IV.	B	Sept. 12, 1912	Aug. 2, 1913	325	4,538.2	358	12.7	2,485	10
b V.	A	Jan. 20, 1913	Aug. 28, 1913	219	8,981.9	252.53	35.5	2,122	15
c III.	3	Mar. 28. 1913	June 1, 1913b	70	404.2	33	12.3	251	50

b Still running on this campaign.

c These two runs are really parts of the same campaign. The converter was shut down on Dec. 25, 1912, to change the tuyeres for experimental purposes.

During October, 1913, all of the Class IV or 12-ft. converters were abandoned, the plant reaching that point in reconstruction where the new 20-ft. converters were used entirely. All the converters referred to in the preceding table as operating in October, 1913, were abandoned while still in operating condition.

If it is desired to learn the performance of the converters listed above in terms of matte treated, the "Tons Cu" should be multiplied by 3.2. The result will be approximately the tons of matte treated.

We believe these figures show greater duty per lining than other basic-lined converters, and we attribute it primarily to the monolithic magnetite lining, properly handled. We even believe that it will be possible to use a cheaper original lining than the magnesite and have made some experiments along that line, but it has not yet been demonstrated to our satisfaction.

The Great Falls Flue System and Chimney.

BY C. W. GOODALE, BUTTE, MONT., AND J. H. KLEPINGER, GREAT FALLS, MONT.

(Butte Meeting, August, 1913)

I. INTRODUCTION.

In the summer of 1909 the Boston & Montana reduction department of the Anaconda Copper Mining Co. completed a new flue system, at a cost of about $1,100,000, and as this includes the largest and highest chimney in the world, and some new features of flue construction, a description will be of interest.

At the Spokane meeting of the Institute in September, 1909, a brief mention was made of this flue system, in discussing Mr. Brunton's presidential paper on the Progress of Metallurgy in the West, and a promise was given that when a long enough time had elapsed, so that operating results could be given, a paper would be offered.

The smelting plant includes blast furnaces, MacDougall roasters, gas-fired reverberatories, and converters.

II. CHARACTER OF THE ORE.

The ore treated comes from Butte, where two classes are made in mining, the first class going to the blast furnaces, and the second to the concentrator. Partial analyses follow:

	Per Cent. Cu	Oz. Ag	Oz. Au	Per Cent SiO₂	Per Cent. Fe	Per Cent. S	Per Cent. Al₂O₃	Per Cent. CaO
First class........	6 06	2 0	0.015	51.2	13.6	17.3	8.1	0 30
Second class..	3 55	1 26	0 008	58 5	9.4	11.6	11 7	0.10

The copper-bearing minerals include chalcocite, enargite, bornite, covellite, tetrahedrite, tennantite, and chalcopyrite,—but the most abundant is chalcocite. Sphalerite and galenite also are found, and in some of the Butte mines these minerals occur in considerable quantities. Pyrite is a very abundant mineral, ranging from 20 per cent. of the weight of second-class ore to 30 per cent. of the first class. The principal country rock of the Butte district is quartz-monzonite, and the gangue of the copper-ore deposits is made up of quartz and highly altered monzonite.

The following analysis will show the composition of the ore and of the unaltered country rock:

	ORE Per Cent.	QUARTZ-MONZONITE [1] Per Cent.
Cu......	3 25	....
As......	0.37	....
Sb......	0.025	....
S........	11.12	....
Fe......	9.30 Fe₂O₃	1.96
	FeO	2.83
SiO₂..	58.45	64.03
Al₂O₃..	11.90	15.58
CaO	0.15	4.20
MgO..	0.25	2.15
K₂O..	2.34	4.11
Na₂O..	0.11	2.76
TiO₂..	....	0.60
P₂O₅..	....	0.18
MnO..	....	0.11
BaO..	....	0.07
SrO..	....	0.04
H₂O below 110° C..	0.02	0.20
H₂O above 110° C..	0.15	0.73

The sample of ore taken for this analysis covers about 90,000 tons of second-class or concentrating ore, received in Great Falls during the month of March, 1913, from the Mountain View, Pennsylvania, Leonard and

[1] Walter Harvey Weed, Geology and Ore Deposits of Butte, Mont., *Professional Paper No. 74, U. S. Geological Survey*, p. 35 (1912).

Tramway mines, and may therefore be taken as representative in character. Leaving out the copper, arsenic, antimony, sulphur and iron, we have:

	Per Cent.
SiO_2...	58 45
Al_2O_3......	11 90
CaO....	0.15
MgO..............	0.25
K_2O......	2 34
Na_2O....	0 11
H_2O above 110° C.....	0 15
	73.35

From this it is seen that the gangue minerals of the ore amount to 73.35 per cent. of its weight. Recalculating this as 100 per cent., we have the following approximate composition of the gangue:

	Per Cent.
SiO_2....	79 70
Al_2O_3	16 22
CaO...........................	0.20
MgO.................................	0.34
K_2O......	3.19
Na_2O	0 15
H_2O above 110° C.....................	0 20
	100 00

From the concentrator the product of the coarse jigs, amounting to about one-third of the entire output of the mill, goes to the blast furnaces, but the fine concentrate goes to MacDougall roasters, and thence into the reverberatories. Partial analyses of these products are given below:

	Per Cent. Cu	Oz. per Ton		Per Cent. SiO_2	Per Cent. Fe	Per Cent. S	Per Cent. Al_2O_3	Per Cent. CaO
		Ag	Au					
Coarse concentrate	12.16	3.9	0.019	20.7	25.9	33.8	4.4	0.3
Fine concentrate..	8.84	3.0	0.021	18.1	28.5	35.9	5.3	0.5

Tables I and II give the results of screen tests on first-class ore, on fine concentrate, and on flue dust, from the main flue chamber.

The test on the flue dust shows that all of it is finer than 0.5 mm., and when we consider the tests on the ore and find that from 7 to 10 per cent. is less than 0.5 mm. in size, it is apparent that much fine material would inevitably pass out of the blast furnaces into the flues.

In the case of the MacDougall roasters, a considerable production of flue dust would naturally be expected, as they treat fine concentrate, 55 per cent. of which will pass through a screen with 0.5 mm. holes.

TABLE I.—*Sizing-Test Results on First-Class Ore.*

Screen Sizes		Pennsylvania Mine			Mountain View Mine			West Colusa Mine		
Mm.	Mesh.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.
On 50.8........		42.80	42.80	4.78	33.09	33.09	5.89	45.00	45.00	8.17
On 45.3........		1.75	44.55	4.58	1.39	34.48	3.42	1.34	46.34	7.38
On 32.0........		7.97	52.52	4.30	7.53	42.01	4.88	6.42	52.76	8.44
On 22.6........		7.78	60.30	4.15	8.17	50.18	4.84	6.48	59.24	8.09
On 16.0........		5.82	66.12	4.21	6.63	56.81	5.25	5.06	64.30	8.88
On 9.51........		7.13	73.25	4.18	8.36	65.17	5.36	6.24	70.54	9.07
On 5.66........		5.21	78.46	4.38	6.30	71.47	5.27	4.74	75.28	9.57
On 4.00........		2.99	81.45	4.45	3.78	75.25	5.17	2.98	78.26	9.75
On 2.38........		3.28	84.73	4.63	4.24	79.49	5.11	3.56	81.82	9.53
On 1.41........		3.22	87.95	4.32	4.12	83.61	4.64	3.85	85.67	8.59
On 0.841.......		2.81	90.76	4.11	3.70	87.31	4.53	3.64	89.31	9.02
On 0.500....50...		2.24	93.00	4.12	2.92	90.23	4.65	2.89	92.20	8.87
On 0.350....90...		0.41	93.41	4.49	0.74	90.97	5.10	0.61	92.81	9.48
On 0.166....200..		2.12	95.53	4.88	2.64	93.61	5.66	2.19	95.00	11.15
On 0.070....200..		1.05	96.58	5.70	1.70	95.31	6.81	1.35	96.35	12.15
Through 0.070....200..		3.42	100.00	3.89	4.69	100.00	4.92	3.65	100.00	7.93
Total........		100.00	......	4.51	100.00	.....	5.32	100.00	.	8.62

TABLE II.—*Sizing-Test Results on Fine Concentrates Fed to the MacDougall Roasting Furnaces and on Main-Chamber Flue Dust.*

Screen Sizes		Fine Concentrates to MacDougalls			Main-Chamber Dust					
					South End of Chamber			North End of Chamber		
Mm.	Mesh.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.	Individual Per Cent.	Cumulative Per Cent.	Assay Cu Individual Per Cent.
On 2.83		9.6	9.6	13.85						
On 2.38		1.5	11.1	14.10						
On 2.00		0.9	12.0	12.50						
On 1.68		3.4	15.4	10.85						
On 1.41		3.3	18.7	9.50						
On 1.19		4.0	22.7	8.80						
On 1.00		2.0	24.7	7.70						
On 0.841		6.6	31.3	7.40						
On 0.707		5.1	36.4	6.85						
On 0.595		2.8	39.2	6.55						
On 0.500		6.0	45.2	6.75						
On 0.350	50	5.4	50.6	6.40	0.5	0.5				
On 0.196	90	21.3	71.9	7.08	2.8	3.3		0.1	0.1	
On 0.070	200	12.2	84.1	8.45	15.8	19.1		1.9	2.0	
Through 0.070	200	15.9	100.0	9.75	80.9	100.0		3.2	5.2	
								94.8	100.0	
Total		100.0	....	8.71	100.0		9.86	100.0		6.60

South end of main dust chamber = entrance; north end = outlet.

III. OLD FLUE SYSTEM.

The arrangement of flues in use prior to 1909 is shown in Plate I. Waste gases from the various furnaces entered three main flues, through which they were conducted to the chimney which is 186 ft. high by 20 ft. inside diameter, and stands on a hill, about 1,300 ft. from the main smelter building.   These flues were built about three-fourths underground, the walls being of rubble masonry, and having a 9-in. semi-circular brick arch roof.   The cross-sectional area of flues Nos. 1 and 2 was 119 sq. ft. each; of No. 3 flue 152.6 sq. ft., making a total of 390.5 sq. ft., against a chimney area of 314 sq. ft.

Under normal conditions of operation, flues Nos. 1 and 2 carried the blast-furnace gases and a part of the reverberatory gases.   Flue No. 3 carried the MacDougall and converter gases and the remainder of the reverberatory gases.

Door openings 3 ft. 2 in. by 5 ft., with cast-iron frames and doors, were built in the walls of each flue at irregular intervals of from 100 to 200 ft., for use in cleaning out the dust, which, after accumulating to a depth of several feet, would seriously throttle the draft on furnaces.   No provision was made for removing the dust while the furnaces were in operation, so that it was necessary to shut them down at least once per year in order to clean out the flues.   At such times all men available were pressed into service and the work continued on three shifts, night and day, until it was completed.   The flues were always hot, causing the men to perspire; the dust stirred up in shoveling was very irritating to the skin, and the work was in general always a source of annoyance to the management and of dissatisfaction among the men.

In order to hasten the work, the dust was simply wheeled to the outside of the flue and dumped in piles on the ground.   Later, it was taken to the furnaces, involving extra handling and expense.

The so-called dust chambers near the MacDougall furnaces and blast furnaces were little more than header flues.   Only the heavier particles of dust settled in them, and, while this amounted to a considerable tonnage, it was only a small percentage of the dust produced by the furnaces.   The same may be said of the main dust chamber, through which the gases from the MacDougall furnaces and converters passed. Each of these chambers was so arranged that the dust could be drawn out while the furnaces were in operation.

The dust from the blast-furnace chamber was drawn into wheelbarrows, elevated and wheeled to a bin, from which it could be drawn into reverberatory charge cars.   Dust from the MacDougall-furnace flue was

Chimney
185' 0" high
20' 0" dia. inside
Area 314 sq. ft.

Area 119 sq. ft.
Flue No. 2

Elev. 3722.5

159' 0" Common
brick

Elev. 3536.5
27' 0" Base

CROSS SECTION OF FLUES

21' 0"

10' 6"

8' 0"

10' 0"

21' 0"

10' 0"

8' 0"

Area 119 sq. ft.
Flue No. 1

21' 0"

12' 0"

8' 0"

Area 152.5 sq. ft.
Flue No. 3

1270' 5" Approx.

Push Car

2' 6"

30'-11"

10' 25#
I B

25' 0"

1¼" Tie rods.

31' 0"

Rails 30' 0" long

El. 3392.25

Brick

13"

CROSS SECT. OF STONE DUST CHAMBER

Blast Furnace
Dust Chamber

Retaining Wall

El. 3382.42

El. 3317.0

El. 3310.0

Flue No. 1

Flue No. 2

Flue No. 3

12' 31.5# I Beam

10' 13¼"

10' 25# I Beam

SECT. OF MAC DOUGAL FURNACE FLUE

N

C

C

C

Dampers closed under normal conditions

Cross flue Chamber

SECTION B-B

A

A

A

Blast Furnace

3' 0"

4'

El. 356.25

El. 3278

Door to
draw
dust

12"

El. 3372.5

Blast furnace
dust chamber

158' 6"

Dampers

B

Blast furnace
dust chamber

545' 0" Approx.

91' 6"

El. 3392.25

Mac Dougal
furnace flue

SECTION A-A

Track

Turntable

Elevator

Main dust Chamber

Dust
chamber

24' 0" Track

90' 0"

Building Line

15' 0" dia.

Mac Dougal
furnace flue

Uptake

29' 2"

Reverberatory
furnace flue

Settlers

Elevator

Valve
boxes

Converters

Platform

Mac Dougal
furnaces

Blast furnaces

17' 5"

141' 0"

Mac Dougal fce.
Converter flue
El. 3314.0
Converter
El. 3271

Reverb. fce. Charging Floor

Building Line

275' 0" Converter flue

Reverberatory furnace

PLATE I.—OLD FLUE SYSTEM.

drawn into regular calcine cars, pushed by hand to a transfer car and delivered to reverberatory charge hoppers. Dust from the main chamber was drawn from chutes on either side into push cars which were then lowered on an elevator to the reverberatory charge-floor level and pushed by hand to a transfer car and delivered to the charge hoppers. This main chamber was so wide that only a part of the dust could be drawn out, making it necessary to shovel a large amount of it in order to give the chamber a thorough cleaning out.

From this it will be seen that, not only were the flues and dust chambers on the old flue system inadequate to settle and collect the dust being produced by the furnaces, but the method of handling the dust was crude and expensive.

Numerous observations were made to determine the draft, temperature and velocity in the old flues. The results varied widely, depending upon the number of furnaces in operation, the atmospheric conditions, and the amount of dust in the flues; but the following table may be taken as a fair average under normal operations.

	POINT A—NEAR DUST CHAMBER			POINT B—NEAR BASE OF CHIMNEY		
FLUE No.	Draft Reading In. Water	Temperature F.°	Velocity in Feet per Min.	Draft Reading In. Water	Temperature F.°	Velocity in Feet per Min.
1	0.97	511	2230	0.61	476	2330
2	1.06	555	1730	0.33	516	1650
3	0 66	467	2840	0.40	458	2500

Temperature of atmosphere 39° F.; barometer, 26.8.

During a 21-day test on No. 3 flue gases, observations were taken every hour on the velocity, temperature and draft in each flue, at about the location of the point C, Plate I, with results as follows:

FLUE No.	Draft Reading In. Water	Temperature F.°	Velocity Ft. per Min.	Volume Gas Calculated to 400° F. Cu. Ft. per Min.
1	0.883	490	2,470	247,800
2	0.96	555	1,750	172,000
3	0 624	465	3,000	377,000
TOTAL	....	....	....	796,800

Average temperature of atmosphere, 39° F.; average barometer, 26.94.

The total draft energy in the old flue system, when all flues were in operation, was used up in about the proportion indicated in the following tabulation:

	LOSS IN IN. WATER		
	Flue No. 1	Flue No. 2	Flue No. 3
Loss from furnace inlet to point A...............	1.53	1.69	1.04
Loss from point A to point B	0.53	0.24	0.78
Loss from point B to top of chimney............	0.40	0.58	0.67
Head equivalent of velocity of escaping gases....	0.23	0.23	0 23
Total theoretical draft..	2.69	2 74	2 72

During conditions described above the gases from four blast furnaces and one reverberatory were being handled jointly by flues 1 and 2, while No. 3 flue was handling gases from 22 MacDougalls, two reverberatories, five converters and three Brückner dryers.

IV. TESTS ON OLD FLUE SYSTEM.

1. Aspiration Tests.

In the year 1900, soon after the capacity of the plant was increased, it became evident that a considerable amount of dust was passing out through the chimney and being lost. In 1903, some preliminary tests were made to determine, if possible, the amount and value of this material. Small quantities of gas were aspirated from the chimney and also from the flues. A large number of samples were taken, the amount of gas handled in each varying from 50 to 1,000 cu. ft. The velocity in the flues at this time was estimated by inserting pieces of newspapers in the flue and noting the time required for them to appear at the top of the chimney. Paper was inserted at different times at various points along the flue and an average of all observations taken. These estimates were later found to be only about 60 per cent. of the true velocity.

On account of the small quantity of gas handled, and the method of sampling, the results were not considered reliable, and in 1904 tests were started on a larger scale.

2. Tests with Steel Settling Chamber.

By means of an exhaust fan, gas was drawn from the flue, through a steel pipe 30 in. in diameter and 105 ft. long. This pipe was considered to be the settling chamber, and was connected to the flue by a smaller

pipe, on the end of which an elbow was so arranged that its open end faced the current of gas in the flue. In all, 31 tests were made with this arrangement, including tests on each of the old flues. The actual continuous running time of each varied from 28 to 42 hr., the average being about 35 hr. The velocity in the settling chamber for a given test was kept as nearly uniform as possible, but was varied for the different tests from 30 to 300 ft. per minute. After each test run was finished, the settling chamber was cleaned out, the dust weighed, sampled and assayed, and a new test started.

While valuable information was obtained from these tests, the results were still unsatisfactory, owing to the great amount of moisture which condensed in the pipe. The form and construction of this type of settling chamber being so radically different from the construction that would be necessary in a chamber of practical size made it hard to draw any conclusions as to what would be required to collect the dust properly from all the smelter gases.

In order, therefore, to make tests under conditions which would more nearly approximate actual working conditions, and at the same time to try out several suggested plans for hastening or increasing the settlement of dust, an experimental flue was constructed of brick. The general plan and arrangement of this flue is shown on Plate II. It was 4 ft. wide by 4 ft. 6 in. high and 304 ft. long, inside dimensions. As in the case of the steel pipe, the gases were drawn through this flue by means of an exhaust fan.

The inlet was through a steel pipe 22 in. in diameter. This pipe entered the old flue through the side wall and was fitted with an elbow at the inner end, the opening in which faced the current of gas at a point near the center of the flue.

With this general arrangement, 10 tests were made, numbered from 32 to 41 inclusive. The first eight of these, 32 to 39 inclusive, were made on gases drawn from No. 3 flue. During the last two, gases from No. 1 flue were tested, No. 2 flue being closed.

Before discussing these tests in detail, it may be of interest to explain the general method of conducting them, together with a description of the apparatus used for determining the amount and character of the gases handled and other physical conditions.

Testing Apparatus Used.

Ordinary mercury thermometers were used for determining temperatures. Those used on the experimental flue were regular glass-stem thermometers, 12 to 16 in. in length, and were inserted through a hole in the roof, and held by a rubber stopper. Those used on the main flues
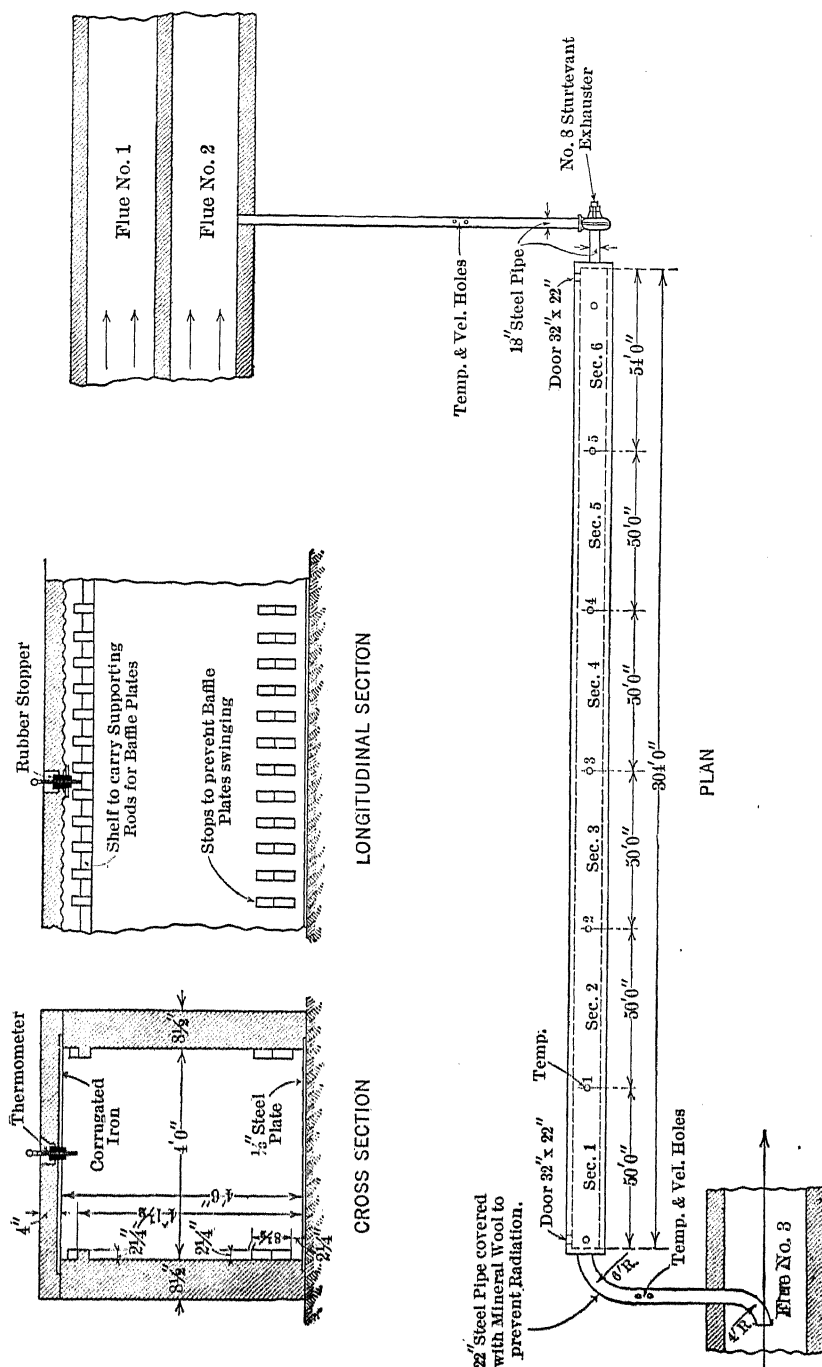
PLATE II.—EXPERIMENTAL BRICK FLUE.

were long-stem mercury flue thermometers, the stems being about 36 in. long.

In measuring velocities, Pitot tubes were employed, a special type being developed for this work. In recent years this instrument has come into such general use, and its principles have been so thoroughly discussed and explained, that any consideration of this phase of the subject seems unnecessary for the purposes of this paper.

While the underlying principle is the same in all its forms, there are almost as many types of the instrument as there are experimenters using it. It is a well recognized fact that the dynamic- or impact-pressure element of the tube is very easy to construct and manipulate, and that it will accurately indicate the velocity of a moving stream or current of fluid, provided the open end squarely faces the direction of flow. The static-pressure element is the one which requires careful attention. To measure the static pressure properly, a form of tube must be used which will not be affected by the impact or velocity pressure of the moving stream of fluid.

Several types of static-pressure end were tested by us. Air from a chamber in which the pressure was maintained practically constant was passed through a well-rounded orifice into a pipe 12 in. in diameter and about 40 ft. long.

Pitot tubes with different forms of static-pressure end were placed in this pipe, and the readings from the static-pressure side were compared with each other and also the volume of air indicated by each type of Pitot tube was compared with the volume calculated as passing through the orifice.

The form of pressure end finally adopted by us is shown in Plate III. It consists of an oval brass plate 2.5 by 4 in. by about $\frac{1}{16}$ in. thick, having the edges beveled on top. In the center and on the long axis of this plate, a slot is cut $\frac{3}{64}$ in. wide by 2 in. long. This slot communicates with a chamber in a light brass casting which is riveted or brazed to the top of plate as shown. The tube leading from this chamber to the manometer may be of any suitable form. For use in small flues or pipes all parts of the instrument should be kept as small and light as possible, consistent with strength, so as to offer the least possible resistance to the flow of gases. When placed in a jet of air from a nozzle in the side of a chamber in which the pressure was 12 lb. per sq. in. gauge, which corresponds to a velocity of about 850 ft. per second, this form of pressure end shows no appreciable reading on the manometer.

Assuming that the pressure energy under such conditions is all converted into velocity, the conclusion is reached that this type of pressure end will correctly indicate the static pressure and is not affected by velocity. It is to be understood, of course, that the tube is to be so placed

that the direction of flow of the moving stream of gas or air is parallel to the bottom plane of the plate and to the slot in the plate.

Other types of pressure end tested, when similarly placed in an air jet, showed readings on the manometer of from minus 84 in. of water, for a straight open-end tube, to plus 3 in. of water, for a tube having on its end a thin circular plate 4 in. in diameter.

The type of pressure end designed and used by Capt. D. W. Taylor, naval constructor, in his Experiments with Ventilating Fans and Pipes,[1]
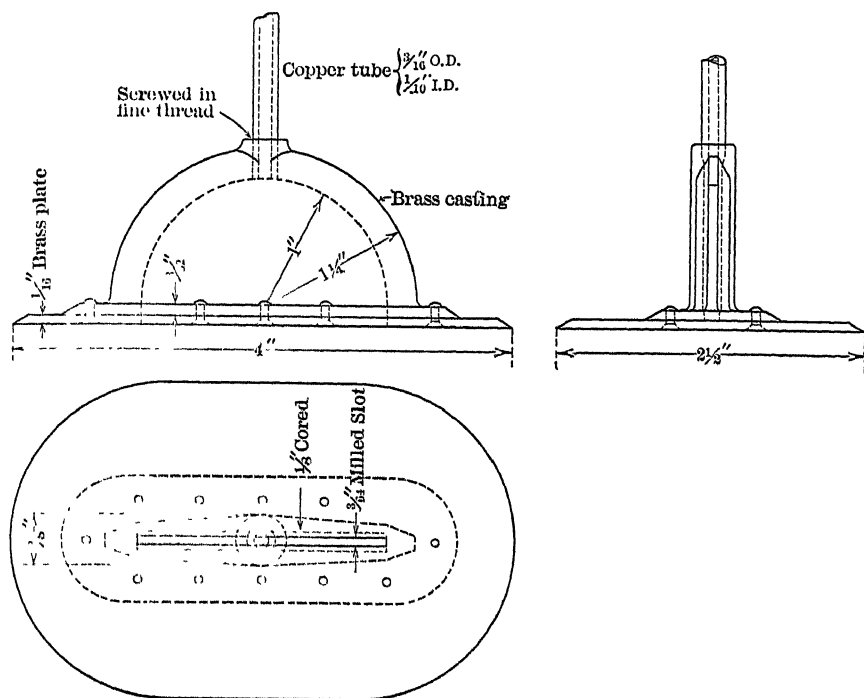
PLATE III.—PITOT TUBE, STATIC PRESSURE END.

was among those tested and was found to be very accurate and, in fact, has been frequently used by us in determining the velocity of air in pipes, etc., but this type was not considered as well adapted for work in connection with flue gases as the type which we used.

In connection with the work of testing Pitot tubes, we also made some observations to determine the average velocity over the whole sectional area of a pipe, and the percentage which this velocity is of the velocity at the center of pipe. The pipe was assumed to be divided into a number of concentric rings, the area of each being multiplied by the average velocity in it as determined by the Pitot tube. These observations indicated

that for a 12-in. pipe the average velocity of the whole cross-section is 89.6 per cent. of the central velocity.

This value, however, was seldom used for calculations on our tests. Instead of taking one reading at the center of a pipe or flue, we used a Pitot tube having five impact points and one static-pressure point,
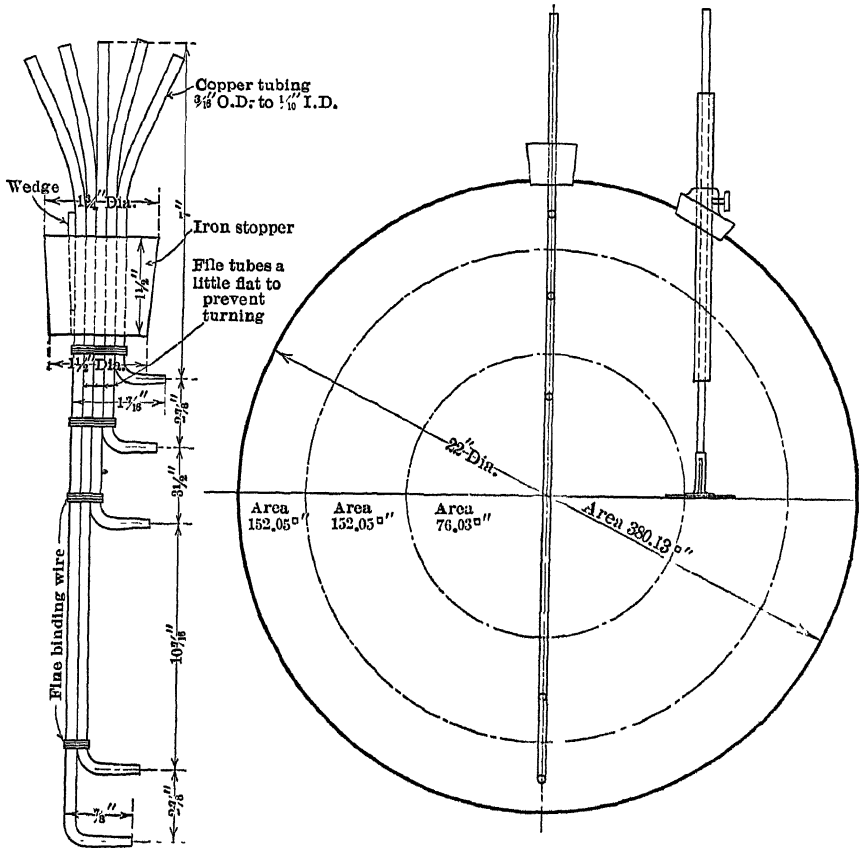
PLATE IV.—IMPACT END OF PITOT TUBE FOR 22-IN. PIPE.

the assumption being that the static pressure was practically the same at all points in the flue or pipe. Plate IV shows the arrangement and construction of the tube as designed for use in a 22-in. pipe. For the main flues, 0.25-in. gas pipes of the required length, having a 90° bend at the lower end, were used. These were not nested together as in the case of the tube for pipe work, but were inserted independently at points where it was thought a good average velocity would be obtained. Plate XXXVIII shows the location and arrangement of these tubes.

For use in connection with this multiple-point Pitot tube, we designed and, so far as we know, put into use for the first time what we have called

an automatic averaging manometer. This instrument is shown in Plate V, Fig. 1. The construction of this manometer is clearly indicated in the illustration and needs little further explanation. The first instrument was made of cast iron, but this soon corroded and filled the liquid with rust, which so clogged and clouded the glass tube that accurate readings could not be made, so that it was necessary to construct it of brass. Each of the chambers in the main body of the instrument must be of the same and of uniform diameter. The diameter of these chambers is made large in proportion to the diameter of the glass tube on which readings are taken, so that, within reasonable limits, the variation in level in these chambers may be neglected and the total reading obtained on one leg of the instrument. The various points of the Pitot tube were connected to the manometer by means of pure gum-rubber tubing, care being taken to see that good joints were made. This instrument is easy to operate and requires little attention. It is defective in that the velocity head which it indicates is the arithmetical average of velocity heads at the several points of the Pitot tube, while the true average velocity varies as the square root of the heads. The difference in these two values, however, is small for velocities such as those with which we were dealing, and we thought it more accurate to use five points in this way than to use one central point and apply an arbitrary factor for obtaining an average.

To calculate the velocity from the reading given by a Pitot tube, the general formula

$$V^2 = 2gh$$

may be used for velocities up to 5,000 or 6,000 ft. per minute, unless extreme accuracy is desired. For higher velocities or for very accurate work a more exact and complicated formula may be used.[1]

In the above formula

V = velocity in feet per second.
g = acceleration due to gravity = 32.16.
h = the height of a column of air or gas, the velocity of which is being measured, equivalent in weight to a column of like sectional area of the liquid used in the manometer, and of height equal to the manometer reading.

In our work, we used grain alcohol in the manometer. Being less dense, this liquid works more freely than water in a small glass tube and for the same reason gives a larger reading which is an advantage where low velocities are being measured.

In connection with single-point Pitot tubes, and for taking draft readings, frictional resistances, etc., we used a manometer similar to that illustrated by Fig. 2, Plate V. It will be noted that one leg of this

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[1] See article on Experiments on Ventilating Fans and Pipes, by D. W. Taylor, Transactions Society Naval Architects and Marine Engineers, Vol. XIII (1905).
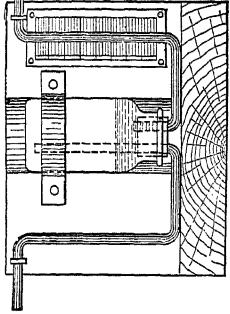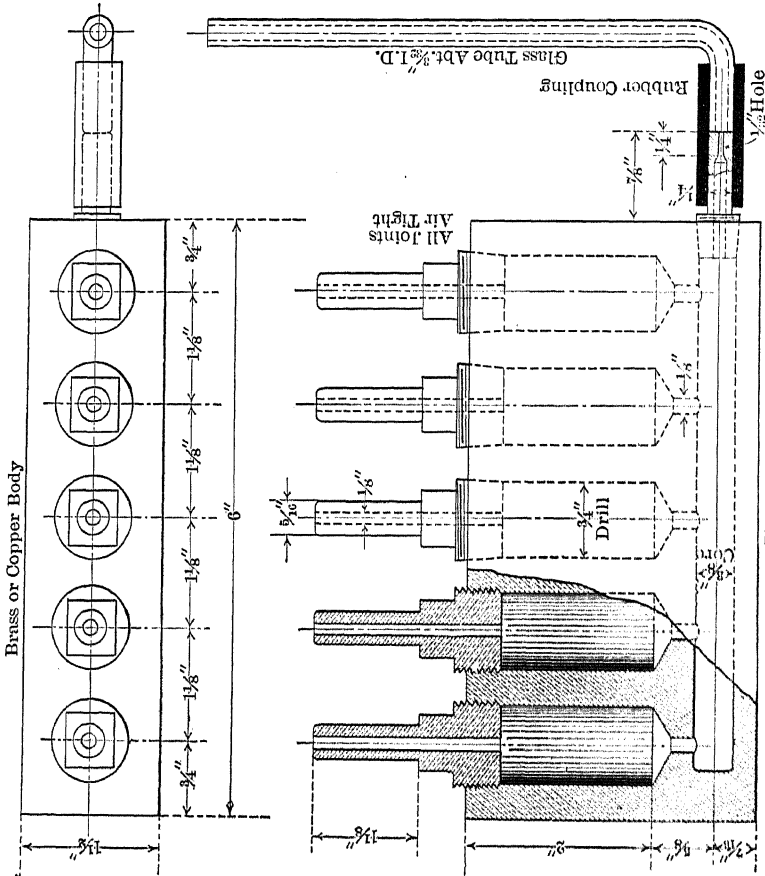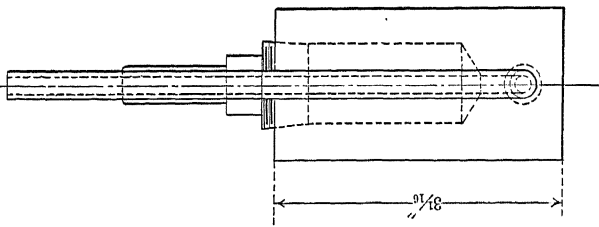
PLATE V.—AUTOMATIC AVERAGING MANOMETER.

manometer is made larger in diameter, so that the movement in it may be neglected, the total reading being obtained on the small leg.

Barometric pressures were obtained from a standard recording aneroid barometer.

3. Test with Experimental Brick Flue.

Referring again to Plate II., showing general arrangement of experimental brick flue: Thermometers were inserted through the roof of flue, as shown, at intervals of approximately 50 ft. Temperatures as well as velocities were taken in the inlet and outlet pipes at points indicated, as well as in the main flue. After a test was started the exhaust fan was kept in continuous operation, day and night, until it was completed, observations of temperature, velocity, etc., being taken every hour.

For the purpose of determining the rate of settlement as well as the character of the dust at different points in the flue, it was divided into six sections, the first five of which, numbered from the inlet end, were 50 ft. long each, the last or sixth section being 54 ft. long. At the end of each test the dust was cleaned out, weighed and sampled by sections.

During test No. 32, there were no baffles or obstructions of any kind placed in the flue, the full cross-sectional area being clear and open as built.

During test No. 33, surface plates similar to Freudenberg plates[1] were hung in the flue for a distance of 170 ft., beginning about 8 ft. from the inlet end. The number and arrangement of these plates is shown on Plate VI, Fig. 1. These plates were thin copper sheets such as are used in our electrolytic refinery for starting sheets. It will be noted that in the space filled with sheets, a baffle plate 8 in. high was placed on the floor every 10 ft.

Test No. 34 was with an open flue, the conditions in every respect being as nearly as possible a duplicate of those in test No. 32.

For test No. 35, baffle plates were hung in the flue, the arrangement being as indicated on Plate VI, Fig. 2. There was a total of 184 ft. of the flue filled with these plates, arranged in two divisions.

The first division started 8 ft. from the inlet end of the flue and extended for a distance of 92 ft., or to the end of Section 2. Next there was 100 ft. of clear flue, covering Sections 3 and 4; followed by 92 ft. of baffles, covering Section 5 and the greater part of Section 6.

These baffles were copper sheets 6.25 in. wide, being sheared from the surface plates used in test No. 33. A clip was attached to the lower end of each plate, through which a rod was passed to engage the stops on flue walls and prevent the baffles from swinging.

In test No. 36, 3⅛-in. baffle plates were used throughout the flue, with the exception of about 8 ft. at each end. The number and arrange-

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[1] For description see Hofman's *Metallurgy of Lead*, Sixth Edition, p. 389.

ment of baffles for a given section was the same as that shown for the
6.25-in. baffles; that is, the free area of the flue was reduced only about
one-fourth instead of one-half.

Test No. 37 was made to determine the effect on the settlement of dust
of expansions and contractions of the gases.   Two bulkheads were built

Fig. 1   Fig. 2

CROSS SECTION THROUGH   CROSS SECTION SHOWING
     SURFACE PLATES   BAFFLE PLATES
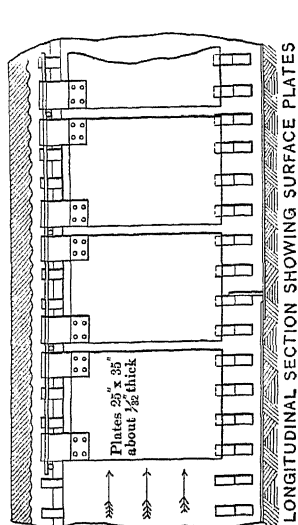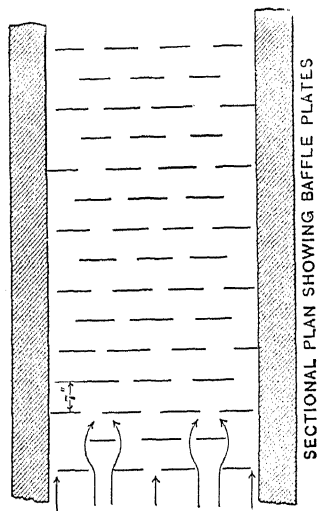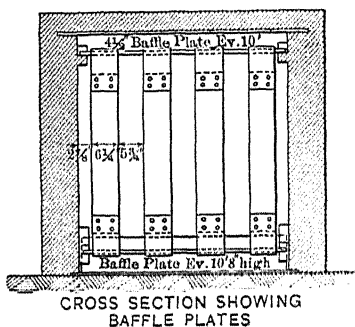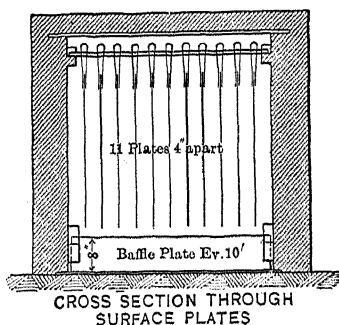
PLATE VI.—EXPERIMENTAL BRICK FLUE BAFFLING ARRANGEMENT.

across the flue at 100 ft. and 200 ft., respectively, from the inlet end.
In the center of each of these bulkheads, a pipe 18.5 in. in diameter was
placed, all as shown on Plate VII.   The inlet pipe, together with these
two bulkhead pipes, made in effect, three expansions and contractions.
Other portions of the flue were clear of any baffles or obstructions.

Test No. 38 was of a preliminary nature, for the purpose of determining
the effect of wire baffles[1] in the settlement of flue dust.   This method of

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[1] See Hofman's *Me'allurgy of Lead*, Sixth Edition, p. 392.

settling flue dust was invented[1] by Dr. Bernhard Rosing, of Friedrichs-hütte, near Tarnowitz, Prussia, Germany. It was used with good success in settling the dust produced by a smeltery at Tarnowitz, but, up to this time, had not been used on a large scale in this country. As originally designed by Dr. Rosing and used in Germany, the wires were hung in

CROSS SECTION AT CONTRACTED AREA
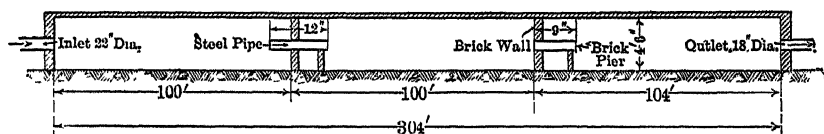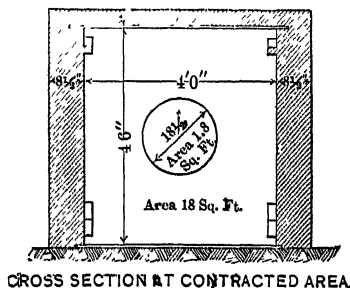
PLATE VII.—ARRANGEMENT OF EXPERIMENTAL FLUE FOR EXPANSION AND CONTRAC-TION TESTS.

small chambers through which the gas passed parallel to the wires, but in our work we passed the gas through them at right angles to their length. Two 50-ft. sections of the flue, viz., No. 3 and No. 6, were hung with No. 7, B. & S. gauge copper wires, which extended from near the roof to near the floor of the flue and were supported by common poultry net-ting. They were spaced at from 2 to 2.25 in. centers.

For test No. 39, which was the longest test run made, the flue was hung with copper wires throughout, with the exception of about 9 ft. at each end, making a total of 286 ft. of wire baffles. The wires were supported in the same way as for test No. 38, and were spaced the same, at 2 to 2.25 in. centers.

For tests Nos. 40 and 41, which were made on gases drawn from No. 1 flue at a time when it was handling gases from blast furnaces only, the number and arrangement of wire baffles was the same as for test No. 39.

During all tests, the 22-in. inlet pipe was boxed in and lagged with

1 Flue-Dust Collector, U. S. Letters Patent 432,440.

TABLE III.—*Summary of Tests Made in Rectangular Brick Flue.*

Test No	Duration of Test Hr.	Volume of Gas Handled per Test Cu. Ft. at 400° F.	Total Dust Collected Lb.	Dust per Million Cu. Ft. Gas at 400° F. Lb.	Average Assay of Dust Per Cent. Cu	Average Assay of Dust Silver Oz. per Ton	Average Velocity in Ft. per Min. Experimental Flue	Average Velocity in Ft. per Min. Main Flue	Average Velocity in Ft. per Min. Inlet Pipe to Exp. Flue	Average Temperature F°. Main Flue	Average Temperature F°. Inlet End Exp. Flue	Average Temperature F°. Outlet End Exp. Flue
32	138.8	89,090,000	3,429.0	38.49	7.69	3.93	568	2,509	3,986	368	362	250
33	143.3	90,016,000	4,336.0	48.17	7.35	3.944	561	2,570	3,987	406	369	254
34	117.6	74,070,000	2,924.0	39.48	9.45	4.716	553	2,590	3,994	389	355	258
35	143.0	42,400,000	4,039.8	95.27	6.83	3.511	266	2,735	1,071	440	373	187
36	163.2	65,540,000	5,064.6	77.25	8.02	4.094	344	2,819	2,740	376	336	187
37	113.9	49,324,000	3,992.3	80.94	7.42	3.5	414	2,803	3,493	491	427	258
38	108.8	53,291,000	4,832.4	90.68	7.75	3.98	477	2,964	3,909	496	444	305
39	491.4	221,039,000	21,825.0	98.69	6.46	3.46	440	3,000	3,069	465	449	311
40	165.4	73,372,000	6,730.4	91.73	6.1	2.99	454	3,084	3,203	549	490	297
41	164.9	71,626,000	18,251.6	254.82	6.7	3.15	470	3,277	3,307	623	540	335

NOTE: Tests 32 to 39 inclusive made on gases from No. 3 main flue. Tests 40 and 41 made on gases from No. 1 flue when handling blast furnace gases only, four furnaces being in operation during each test.

Conditions of Test.

Test No.		AVERAGE NUMBER OF FURNACES DISCHARGING INTO FLUE Blast	MacDougall	Converter	Reverberatory	Brueckner
32	Clear flue, no baffles.		19.0	3.9	1	3
33	Surface plates for length of 170 ft., beginning 8 ft. from inlet end		17.5	5.	1	3
34	Clear flue, no baffles.		18.4	4.6	1	3
35	6.25 in. baffle plates in sections 1, 2, 5 and 6; sections 3 and 4 clear flue.		21.3	5	1	3
36	3.125 in. baffle plates throughout the flue, except about 8 ft. at each end.		20.0	3.6	1.8	1.9
37	Expansions and contractions, three stages.		21.6	5.3	1.8	3
38	Wire baffles in sections 3 and 6, other sections clear flue		22	6	2	3
39	Wire baffles throughout flue except about 9 ft. at each end.		22	4.8	2	3.5
40 and 41	Same as No. 39	4				

mineral wool, so that the gases would be delivered to the experimental flue at as near normal temperature as possible.

Table III gives a general summary of the results of tests 32 to 41 inclusive. In comparing the results of tests in which different kinds of baffles were used, consideration must be given to the fact that in tests 33, 35 and 38, only a part of the flue was equipped with baffles. Direct comparisons may be made among tests 32, 34, 36 and 39. Tests 40 and 41 will be discussed a little later in this paper.

Table IV presents in more detail the results of test 39. It shows the average temperature in each section, as well as the amount of dust collected in each, with the analysis of the same. It will be noted that the amount of copper decreases from 6.4 per cent. in section 3 to 2.6 per cent. in section 4. At the end of this test (No. 39) it was noted that the wires in section 1, and about one-half of section 2, were clean and free from dust. Beginning at the middle of section 2, there was a deposit of dust on the wires, the amount of which gradually increased, the maximum deposit being at about the division line between sections 3 and 4, at which point the overall diameter of wire and dust was about $\frac{5}{8}$ in.

A moderate rapping or shaking of the wires was effective in removing a larger part of the dust adhering to them.

A marked difference was noted in the character of the dust adhering to the wires in the different sections. Up to about the end of section 4, this deposit was of a crystalline character, while, in the following sections, it was of a light amorphous nature.

In addition to the assays shown in Table IV, a more complete assay was made on a combined sample made up from the several sections in proportion to the amount of dust collected in each. This assay was as follows:

Cu, per cent.. 6 2
Ag, oz. per ton... 3.4
Au, oz. per ton... 0.025
Insoluble, per cent... 38.2
SiO_2, per cent... 26.0
Fe, per cent.. 5.7
Al_2O_3, per cent.. 13.3
CaO, per cent.. 0.8
S, per cent... 7.1
SO_3 (free and combined), per cent..................... 14.1

In order to determine the amount of dust escaping from the experimental flue during test 39, a measured quantity of gas was aspirated, two methods of collecting the dust being used. In one of these methods the gas was filtered through a chamber filled with mineral wool, a comparatively large volume being handled, and in the other, it was aspirated through absorption bottles which were preceded by a bulb filled with mineral wool.

TABLE IV. TEST NO. 39.—*General Summary of Observations and Results.*

SECTION OF FLUE	Temperature of Gas F°.	DUST COLLECTED, LB.		POUNDS DUST PER MILLION CU. FT. GAS VOL. AT 400° F.		PER CENT. OF TOTAL DUST COLLECTED		Cu Per Cent.	ASSAY OF DUST	
		Per Section	Cumulative	Per Section	Cumulative	Per Section	Cumulative		SO₃ Free and C'm'd Per Ct	Silver Oz. per Ton
Inlet Pipe	470	....	....	....	....	....	....	....	....	....
No. 1	449	4,404	4,404	19.87	19.87	20.18	20.18	9.3	6 8	4.4
No. 2	418	6,205	10,609	28.06	47.93	28.43	48.61	7.8	10 1	3.8
No. 3	387	6,277	16,886	28.37	76.30	28.76	77.37	6.4	13 0	3.2
No. 4	358	2,557	19,443	11.56	87.86	11.72	89.09	2.6	20 5	2.2
No. 5	332	1,102	20,545	5.06	92.92	5.05	94.14	2.2	24.6	2.5
No. 6	311	847	21,392	3.81	96.73	3.88	98.02	2.1	30 9	3.2
Exstr. & Con. Pipes	283	433	21,825	1.94	98 67	1.98	100.00	1 6	44 3	2 8

GENERAL DATA ON EXPERIMENTAL FLUE:

Duration of test.. 491 hr. 22 min.
Average velocity in inlet pipe.................................. 3,070 ft. per min.
Approximate mean velocity of gases in flue..................... 440 ft. per min.
Average temperature of gases in flue........................... 376° F.
Average temperature of atmosphere.............................. 39° F.
Frictional and other losses in first 100 ft. of flue.......... 0 4 in. of water
Frictional and other losses in second 100 ft. of flue......... 1 2 in. of water
Frictional and other losses in third 100 ft. of flue.......... 0 7 in. of water
Total gas handled per test, volume at temperature in inlet pipe. 238,887,000 cu. ft.
Total gas handled per test, volume at 400° F.................. 221,039,000 cu. ft.
Pounds dust per million cu. ft. gas............................ 98 69

GENERAL DATA ON MAIN FLUE:

Average temperature of gases................................... 465° F.
Average velocity of gases...................................... 3,000 ft. per min.
Effective area of cross-section................................ 135 sq. ft.
Draft reading at point of velocity measurements................ 0 624 in. of water
Cu. ft. gas passing up main flue per min., at observed temperature. 405,500
Cu. ft. gas passing up main flue per min., volume at 400° F.... 377,000
Lb. dust carried by gases in main flue per 24 hr., as indicated by test. 53,580
Average number of furnaces discharging gas into main flue, 22 MacDougalls, 2 reverberatories, 4.8 converters, 3.5 Brueckners.

The dust collected in each case was weighed and analyzed, and the results were found to check very closely.

Based on these tests, it was estimated that, if all gases being carried by No. 3 flue were passed through a wire-baffled chamber similar to the experimental flue, there would be a loss of 7,320 lb. of dust in 24 hr. Therefore,
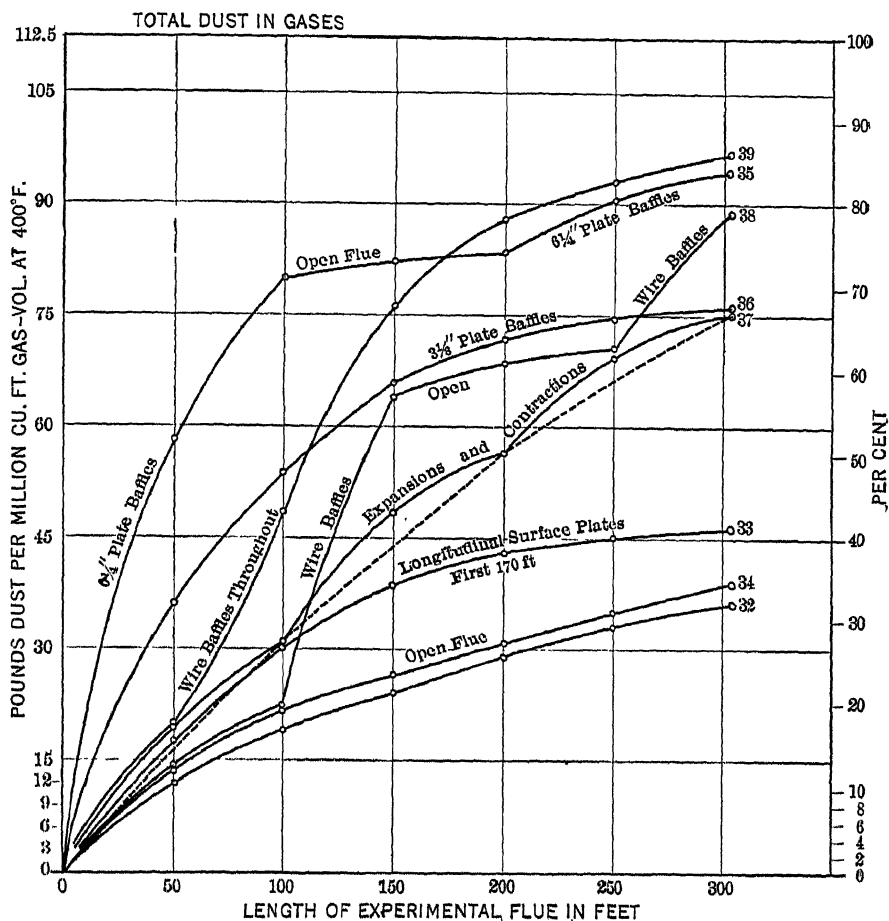
PLATE IX.—RELATIVE EFFICIENCY OF DUST-ARRESTING SCHEMES.

the total amount of dust estimated as passing out through No. 3 flue per 24 hr. would be 53,580 + 7,320 = 60,900 lb., containing 3,580 lb. of copper. It was further estimated, based on the results of test 39, that, if all the gases carried by No. 3 flue should be passed through a wire-baffled flue or chamber under the same conditions of velocity and temperature, the recovery would be 86 per cent. of the dust and 96 per cent. of the copper.

The curves on Plate IX show more clearly than a table the relative merits of the various dust-arresting schemes tested. The horizontal line at top designated as "Total Dust in Gases" is based on the results of test 39. Assuming that this line represents the total average dust in the gases at all times, it may be used as a basis for comparison of the efficiency of the various dust-arresting schemes tried out. From these curves an estimate may also be made as to the probable length of flue necessary to collect a given amount of dust, by the use of any particular scheme, and shows that with an open flue an almost infinite length of flue of uniform cross-section would be required to effect a recovery equivalent to that obtained by the use of wire baffles. It is interesting to note the trend of curves representing tests 35 and 38, in which only certain sections of the flue were filled with baffles. In the open sections for these tests the curves are approximately parallel with the curves for the open-flue tests.

Another observation to be made is that the baffle plates effect a more rapid settlement of dust than other methods; that is, a higher percentage of the total dust is settled in a shorter length of flue, but this is at the expense of draft energy, as will be pointed out later.

The favorable showing made by expansions and contractions in the settlement of dust is of considerable interest and indicates that additional tests along this line might develop some valuable information. The simplicity of the scheme commends it over the more complicated baffling methods, the only serious objection being the draft hindrance which it introduces. It is possible, however, that a greater number of stages, with a lesser percentage of contraction, might give equal efficiency and at the same time offer less resistance to the flow of gas.

Plate X, Fig. 1, shows the relative frictional resistance to the passage of gases per 100 ft. of flue, under the conditions of the test. In comparing these resistances, consideration must be given to the volume of gas handled per unit time for the different tests, the relative amounts of which are shown in Fig. 3, while Fig. 4 shows the relative dust collected per unit volume of gas and Fig. 2 shows the relative draft readings at outlet end of flue, in inches of water. During all of these tests the exhauster was run at practically the same speed, the average being about 1,250 rev. per min.

To determine how the frictional resistance in the wire baffles increased as the test progressed, observations were taken from time to time during test 39, the results of which are presented in Plate XI. The exhauster was run at a practically constant speed of 1,250 rev. per min. The increase in frictional resistance was effective in reducing the volume of gas handled from 9,500 cu. ft. per min. at the beginning of test to 7,100 cu. ft. on the twentieth day. This loss in volume was partially due to a reduction in sectional area on account of the accumulated dust on the floor of the flue, which amounted to nearly a foot in depth in some sections.
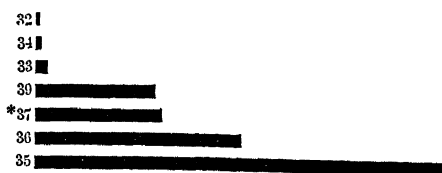
*One Expansion and Contraction,
Ratio of Areas 10 to 1

FIG. 1.—Relative Friction Losses per Hundred Feet of Flue.
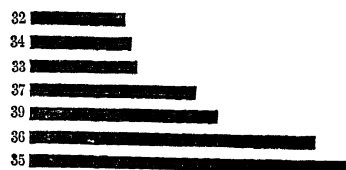
FIG. 2.—Relative Draft Readings at Outlet End of Flue, Inches Water.

FIG. 3.—Relative Volume of Gas per Unit Time.

FIG. 4.—Relative Weight of Dust per Unit Volume of Gas.

Test No.
32. Clear Flue.
33. Surface Plates.
34. Clear Flue.
35. 6.25-in. Baffle Plates.
36. 3.125-in. Baffle Plates.
37. Expansions and Contractions.
39. Wire Baffles.
PLATE X.

Based on velocity determinations made on all main flues during test 39, the total gas discharged from the smelter at that time was as follows:

Flue Number	Average Observed Temp. F°.	Cu. Ft. per Min. at Observed Temp.	Weight Lb.
1	490	273,730	10,200
2	555	203,000	7,080
3	465	405,500	15,520

If all these gases were combined in one main flue, the resulting temperature would be 490° F., at which the volume would be 880,300 cu. ft. per min.

4. Tests on Blast-Furnace Gases.

As already stated, tests 40 and 41 were made on blast-furnace gases only. The experimental brick flue, hung with wire baffles throughout, was used in these tests, the arrangement being similar in every respect to that employed in test 39, with the exception that baffle plates 8 in. high were placed on the floor at 10-ft. intervals, in the last 100 ft. of flue.

Information on two points was sought in these tests. First, a determination of the efficiency of wire baffles in the settlement of blast furnace flue dust; and second, to determine, if possible, what percentage of the briquettes being treated in the blast furnaces was being lost in flue dust.

Previous preliminary tests on blast-furnace gases had indicated that the dust carried by them was more difficult to collect than that contained
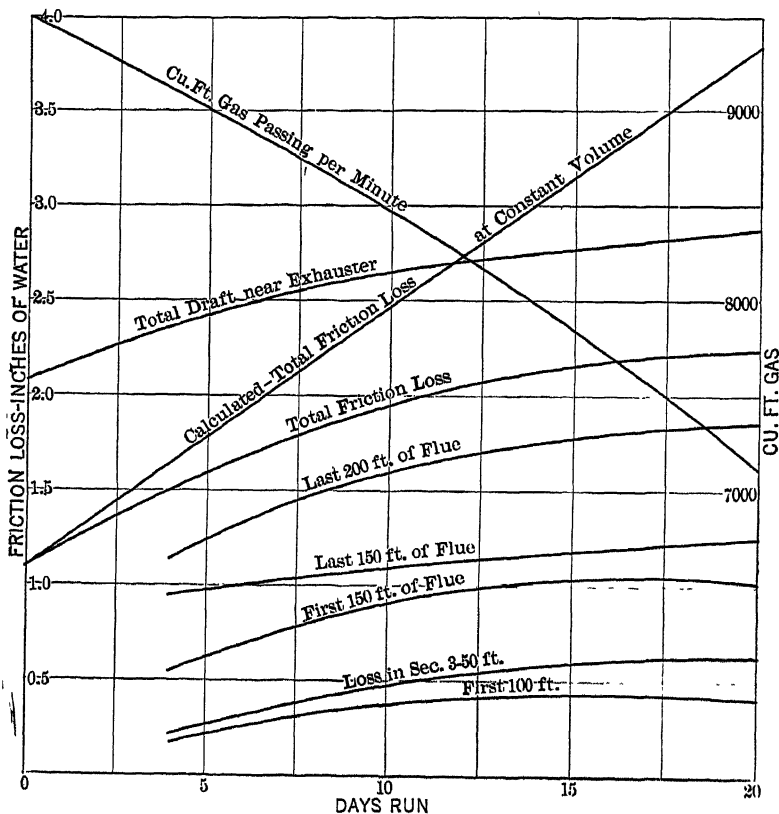
PLATE XI.—FRICTION LOSS THROUGH WIRE BAFFLES, EXPERIMENTAL BRICK FLUE.

in the roaster gases. This conclusion was further confirmed by the present tests. It was evident that some dust was escaping from the experimental flue. To determine the amount so escaping, tests were made similar to those carried on during test 39. Even after filtering the gases through mineral wool, there seemed to be a slight amount of dust still escaping. Assuming, however, that practically all of the dust was recovered in the mineral-wool filter, calculations were made showing the percentage of total dust which was collected in the experimental flue. These results are shown in Table V, in parallel comparison with similar results for test 39, which was made on roaster gases.

TABLE V.—*General Comparison of Results and Conditions for Tests 39, 40 and 41.*

	TESTS		
	39	40	41
Average temperature in main flue, F°... .. .	465	549	623
Average temperature in Sec. 1, Exp. flue, F°. ..	449	490	540
Average temperature in Sec. 6, Exp. flue, F°. . .	311	297	335
Average temperature of atmosphere, F°.... . .	39	45	55
Average velocity in Exp. flue, ft. per min.. .	500	450	470
Dust recovered in Exp. flue, per cent. of total ..	86	81	80
Copper recovered in Exp. flue, per cent. of total..	96	90	91
Average per cent. Cu in dust escaping from Exp. flue	1.65	2 9	2.5
Average per cent. SO_3 in dust escaping from Exp. flue..	45.1	11.7	9.2

It will be noted that the initial temperature of the gases is higher for tests 40 and 41 than for No. 39, which is believed to materially reduce the rate of settlement of dust. While no screening tests were made, the blast furnace dust seemed to be decidedly finer and dryer than the roaster dust, and did not adhere to the wire baffles as readily. The wires were practically clear of dust up to about the middle of the flue, and had only a light deposit for the last, or upper, half of flue. This deposit was readily dislodged by rapping or shaking the wires. The final temperature in tests 40 and 41 was low on account of an excess air leakage into the experimental flue.

Briquettes as Dust Producers.

During test 40 the furnaces were operated under normal conditions, no briquettes being included in the charge. During test 41 the operating conditions were as nearly like those for test 40 as possible, but about 12.5 per cent. of the charge was composed of briquettes. During both tests the furnaces were working well, the tonnage smelted being in general above the average. Practically all briquettes were charged to the furnaces directly from the machine, which was a Chambers auger brick machine with automatic wire cutoff. There was a small quantity which had been on hand for several days, but these were piled so that they would dry out very little. The average amount of moisture contained in the briquettes was 13 per cent., and they were of the following general composition:

Per Cent.

Copper precipitate... 1.31
Slimes... 47.17
Fine concentrate... 51.52

The following table shows the amount of each class of material treated during the tests:

	Test 40		Test 41	
Time represented by material treated, hr................	167.75		166.00	
Actual running time of test, hr....	165.42		164.92	
Average blast pressure, oz........	38.3		41.3	
	Wet Weight Tons	Per Cent. of Charge	Wet Weight Tons	Per Cent. of Charge
First-class ore.....	3,635 5	32.07	3,148.6	26 57
Coarse concentrates.	2,783.9	24.56	2,433.8	20.54
Briquettes...........	.. ..	.. ..	1,477.2	12.46
Converter slag......	1,332.0	11.75	1,362.9	11 50
Lime rock.....	3,585 5	31.62	3,428.7	28 93
TOTALS..	11,336.9	100 00	11,851 2	100 00
Coke....	1,042 7	9.19	1,025 4	8.65

Tables VI and VII show the detailed results of these tests. Based on these results and on results of tests on gases leaving the experimental flue, the total amount of dust in the main flue gases would be as follows:

	Test 40	Test 41
Dust in gases per 24 hr., based on total recovery........	38,530	105,970
Dust in gases per 24 hr., based on escaping solids.......	8,700	26,130
Total per 24 hours..........................	47,230	132,100

The average weight of new cupreous material treated per 24 hr. during test 40 was 918.4 tons. The amount of dust being carried along by the main-flue gases per 24 hr. was therefore equivalent to 2.57 per cent. of this weight.

TABLE VI. TEST 40.—*General Summary of Observations and Results.*

SECTION OF FLUE	Temperature of Gas F.°	DUST COLLECTED, LB.		POUNDS DUST PER MILLION CU. FT. GAS VOL. AT 400° F.		PER CENT. OF TOTAL DUST COLLECTED		ASSAY OF DUST		
		Per Section	Cumulative	Per Section	Cumulative	Per Section	Cumulative	Per Cent. Cu	Per Cent. SO₃ Free and Combined	Ounces Silver per Ton
Inlet Pipe	523									
1	490	1,816.3	1,816.3	24.76	24.76	26.98	26.98	9.0	6 0	3 4
2	430	1,493.6	3,309.9	20.36	45.12	22.19	49.17	8 4	7 1	3 6
3	385	1,229.9	4,539.8	16.76	61 88	18.27	67 44	4.4	10 5	3 3
4	351	975.6	5,515.4	13.29	75.17	14.50	81 94	3 5	17 6	2 0
5	321	737.4	6,252.8	10 05	85.22	10.96	92 90	3 0	20 3	2.3
6	297	449.3	6,702.1	6.12	91.34	6.68	99.58	2 4	19.7	1 8
Exstr. & Con. Pipes	270	28.3	6,730.4	0.39	91 73	0.42	100 00	2 5	17 3	1.8

GENERAL DATA ON EXPERIMENTAL FLUE:

Duration of test .... 165 hr. 25 min.

Average velocity in inlet pipe .... 3,200 ft. per min.

Approximate mean velocity of gases in flue .... 454 ft. per min.

Average temperature of gases in flue .... 380° F.

Average temperature of atmosphere .... 45.5° F.

Frictional and other losses per 100 ft. of wires based on total friction through all wires .... 0 52 in. of water

Total gas handled per test, volume at temperature in inlet pipe .... 83,920,000 cu. ft.

Total gas handled per test, volume at 400° F .... 73,372,000 cu. ft.

Pounds dust per million cubic feet gas 91 73

GENERAL DATA ON MAIN FLUE:

Average temperature of gases .... 549° F

Average velocity of gases .... 3,084 ft. per min.

Effective area of cross-section .... 111 sq. ft.

Draft reading at point of velocity measurements .... 0 67 in. of water

Cu. ft. gas passing up main flue per min., at observed temperature 342,300

Cu. ft. gas passing up main flue per min., volume at 400° F 291,700

Pounds dust carried by gases in main flue per 24 hr., as indicated by test 47,230

Average number furnaces discharging gas into main flue—4 blast furnaces.

TABLE VII. TEST 41.—*General Summary of Observations and Results.*

Section of Flue	Temperature of Gas F°	Dust Collected, Lb.		Pounds Dust per Million Cu. Ft. Gas Vol. at 400° F.		Per Cent. of Total Dust Collected		Assay of Dust		
		Per Section	Cumulative	Per Section	Cumulative	Per Section	Cumulative	Per Cent. Cu	Per Cent. SO₃ Free and Combined	Ounces Silver per Ton
Inlet Pipe	577	....	....	....	....	....	....	....	....	....
1	540	5,559.2	5,559.2	77.61	77.61	30.46	30.46	7.6	4.0	3.2
2	472	3,875.7	9,434.9	54.11	131.72	21.24	51.70	7.4	4.0	3.2
3	425	2,351.8	11,876.7	32.84	164.56	12.88	64.58	6.6	4.4	3.2
4	393	2,121.0	13,907.7	29.61	194.17	11.62	76.20	6.2	4.8	3.0
5	365	2,584.5	16,492.2	36.08	230.25	14.16	90.36	5.7	5.7	3.0
6	335	1,687.6	18,179.8	23.56	253.81	9.25	99.61	4.8	7.5	2.8
Exstr. & Con. Pipes	304	71 8	18,251.6	1.00	254.81	0.39	100.00	2 1	15.9	1 6

GENERAL DATA ON EXPERIMENTAL FLUE:

Duration of test... 164 hr. 55 min.
Average velocity in inlet pipe............................... 3,300 ft. per min.
Approximate mean velocity of gases in flue................... 470 ft. per min.
Average temperature of gases in flue......................... 420° F
Average temperature of atmosphere............................ 55° F.
Frictional and other losses per 100 ft. of wires, based on total friction through all wires ..... 0 45 in. of water
Total gas handled per test, volume at temperature in inlet pipe ..... 86,382,000 cu. ft.
Total gas handled per test, volume at 400° F................. 71,626,000 cu. ft.
Pounds dust per million cu. ft. gas.......................... 254 81

GENERAL DATA ON MAIN FLUE:

Average temperature of gases................................. 623° F.
Average velocity of gases.................................... 3,277 ft. per min.
Effective area of cross-section.............................. 111 sq. ft.
Draft reading at point of velocity measurements.............. 0 68 in. of water
Cu. ft. gas passing up main flue per min., at observed temperature... 363,700
Cu. ft. gas passing up main flue per min., volume at 400° F.. 288,800
Pounds dust carried by gases in main flue per 24 hr., as indicated by test 132,100
Average number of furnaces discharging into main flue—4 blast furnaces.

Applying this percentage to the weight of new cupreous material treated per 24 hr. during test 41, exclusive of briquettes, we obtain 41,484 lb. Deducting this amount from the total dust per 24 hr. as indicated by the test gives a balance of 90,616 lb. of dust per 24 hr. produced by the briquettes.

Since the average dry weight of briquettes charged during the same period was 371,660 lb., the conclusion is reached that 24.4 per cent. of the material of which they were composed was lost in flue dust. An even greater percentage was doubtless carried out of the furnace, some of which was recovered from the dust chamber.

The curves on Plate XII show the rate of deposit and amount of dust collected for these two tests. The curve for test 39 is included for comparison.

The average weight per cubic foot of dust for tests 39, 40 and 41 is as follows:

	Test		
	39	40	41
Section 1.	55.5	60 8	63.3
Section 6..........	30.5	47.6	35.6

V. DESIGN OF NEW FLUE SYSTEM.

Having determined the amount and character of gas to be handled and tried out various styles of baffles for hastening the settlement of dust, the practical problem was then presented of designing a flue system, including a chimney, which would carry all furnace gases, effectively settle the dust and provide for economically handling the same, allow for additional smelter capacity, if desired, and improve the draft conditions existing with the old system.

While all of these improvements were highly desirable and necessary from an operating standpoint, a further problem of considerable importance was the treatment and disposal of waste gases in such a manner as to avoid complaints from surrounding property owners. The advantages are well recognized of discharging smelter gases into the atmosphere at as high an elevation as possible, and for the condensation and settlement of fume, low temperatures are required, thus making a tall chimney a practical necessity.

An important limiting element in the design was the space available for locating a flue and dust chamber, of sufficient size to settle the dust

and at the same time be reasonably accessible for the removal and transfer of dust to the furnaces.

Considering the various methods tested, having in mind efficiency combined with minimum draft hindrance and general practicability, it
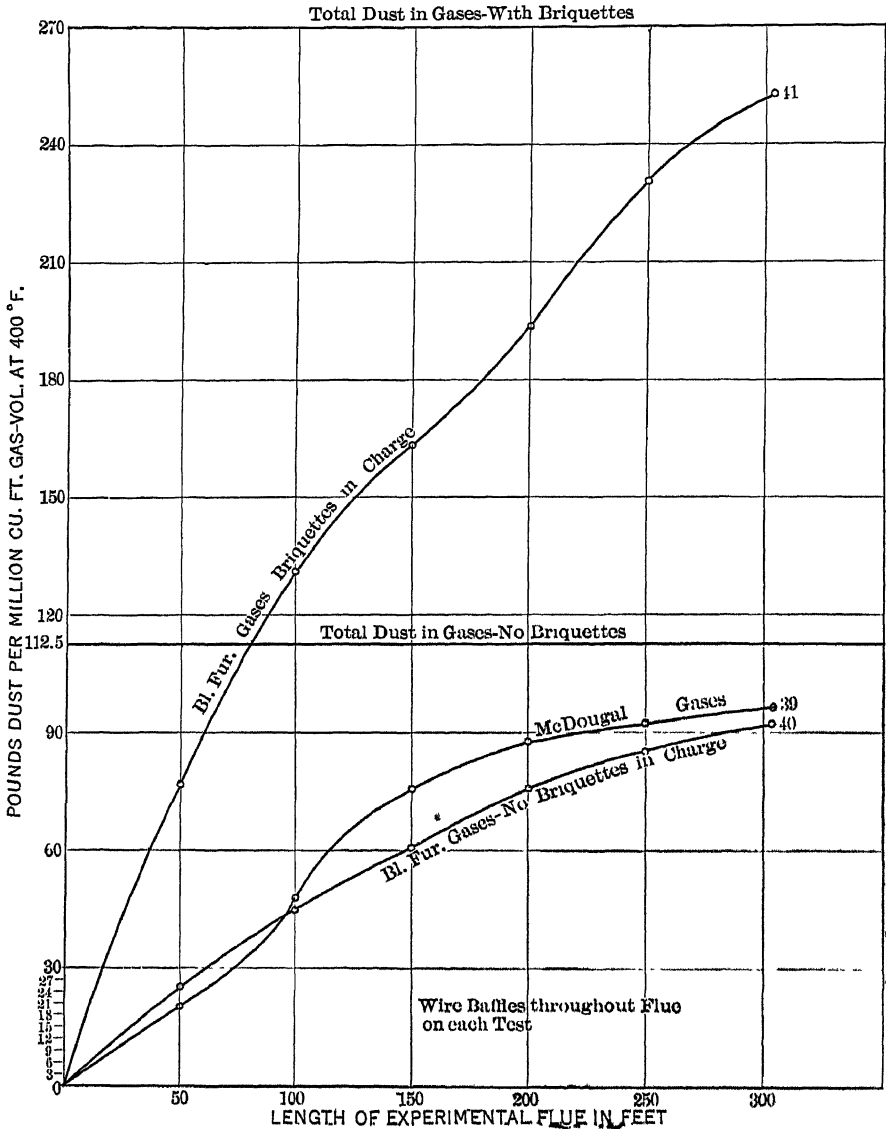
PLATE XII.—CUMULATIVE DUST CURVES.

was decided to use wire baffles and to make the cross-sectional area of the main dust chamber such that the velocity through it would not exceed 450 ft. per min.

Tests for volume and temperature made on individual department flues indicated that, if the plant were operated at full capacity and all gases were combined in one flue near the furnaces, the volume would be about 945,000 cu. ft. per minute at an average temperature of 550° F. In addition to this, an allowance was made for an increase of about one-third in the capacity of the plant, making a total of 1,260,000 cu. ft. of gas per minute based on old draft conditions. To cover an increased volume due to better draft conditions with the new system, we added an additional 25 per cent., making a total estimated volume to be handled of 1,575,000 cu. ft. of gas per minute. At a velocity of 450 ft. per minute this would require a flue having a cross-sectional area of 3,500 sq. ft. The dust chamber, as constructed is 176 ft. wide by 21 ft. high inside, there being 11 bays 16 ft. wide. The size of the flue connecting the main dust chamber to the chimney was established more or less arbitrarily. A velocity of about 1,500 ft. per minute was considered desirable, thus requiring an area of about 1,000 sq. ft. It was therefore decided to build a flue 48 ft. wide by 21 ft. high inside dimensions.

It can scarcely be said that the size of chimney, in so far as the diameter is concerned, was based on any very definite calculation. Some estimates were made on the amount of cold air necessary to bring all flue gases down to about 300° F., in case it was desired to condense the arsenic fumes; an allowance was then made for doubling the capacity of the plant, if required, and the diameter finally established at 50 ft. inside at the top, on the general principle that, in our experience, the greater part of smelter structures had proved too small after a few years' service and had been replaced by larger ones.

In arriving at the height of the chimney, however, a more definite calculation was made. From tests on the old flue system it was estimated that, with the better arrangement of flues contemplated in the new system, there would be required to overcome the frictional and other losses up to the point of entrance to the new dust chamber, a draft energy of 0.97 in. of water. In like manner, from tests on our experimental flue and other observations on the old system, the following total draft requirements were estimated:

	In. of Water
Draft energy required at entrance of new dust chamber	0.97
Draft energy required in wire-baffled dust chamber	0.85
Expansion and contraction loss entering and leaving dust chamber	0.30
Chimney and flue from dust chamber	0.96
Total	3.08

An average temperature of 450° F. was assumed from inlet of dust chamber to base of chimney, the height being 146 ft.

An average temperature of 350° F. was assumed in the chimney; atmosphere temperature 70° F. From these data, and assuming the gases to be air, it is found that the 146-ft. rise in flue from dust chamber to base of chimney will produce a draft energy of 0.78 in. of water. Deducting this from the total of 3.08 in. as found above leaves 2.3 in. to be produced by the new chimney. For an average temperature of 350° in the chimney this would require a height of 515 ft. A height of 500 ft. was therefore decided upon, but the actual height of the chimney as contracted for and built was 506 ft. A question that is sometimes asked is why the extra 6 ft. was added. In answer to this it may be explained that when specifications were made covering the construction of this chimney, only preliminary plans for the flue system had been made. These plans provided for an excavation 6 ft. in depth for the flues, at which elevation the brickwork for chimney was to begin, and, since it was desired to have the chimney 500 ft. above the surface of ground, a height of 506 ft. was specified. Later, these plans were changed so that the bottom of flues as well as base of chimney was at or slightly above the ground line, but the chimney was built 506 ft. high as per original specification.

The location of the chimney near the site of the old one was practically fixed by the topography of the country, while the location and length of dust chamber was largely determined, as already stated, by its accessibility and by the location of other structures and equipment.

Plate XIII shows a general plan and arrangement of the new system. It should be explained at this point that the smeltery end of this plan will soon be a matter of history, since the number, size and location of furnaces will be entirely changed when reconstruction of the plant, which is now well under way, is completed. It does represent, however, the arrangement and conditions under which the new flue system has been working since it was put into operation on June 12, 1909.

In addition to the parts of the system already described, new department flues were built for the blast furnaces and for the MacDougall furnaces. The old converter flue joins the MacDougall furnace flue near its entrance to the main uptake. The system is arranged so that the reverberatory furnace gases can be bypassed around the main dust chamber, or be passed through it as desired.

All parts of the new system are equipped with hopper bottoms up to the dampers at upper end of the main dust chamber, so that the dust can be completely cleaned out while the flues are in operation.

In Plate XIII is shown a cross-section through the blast-furnace department. Each furnace is connected to the flue by means of a steel pipe goose-neck, as shown. The end entering the flue is equipped with
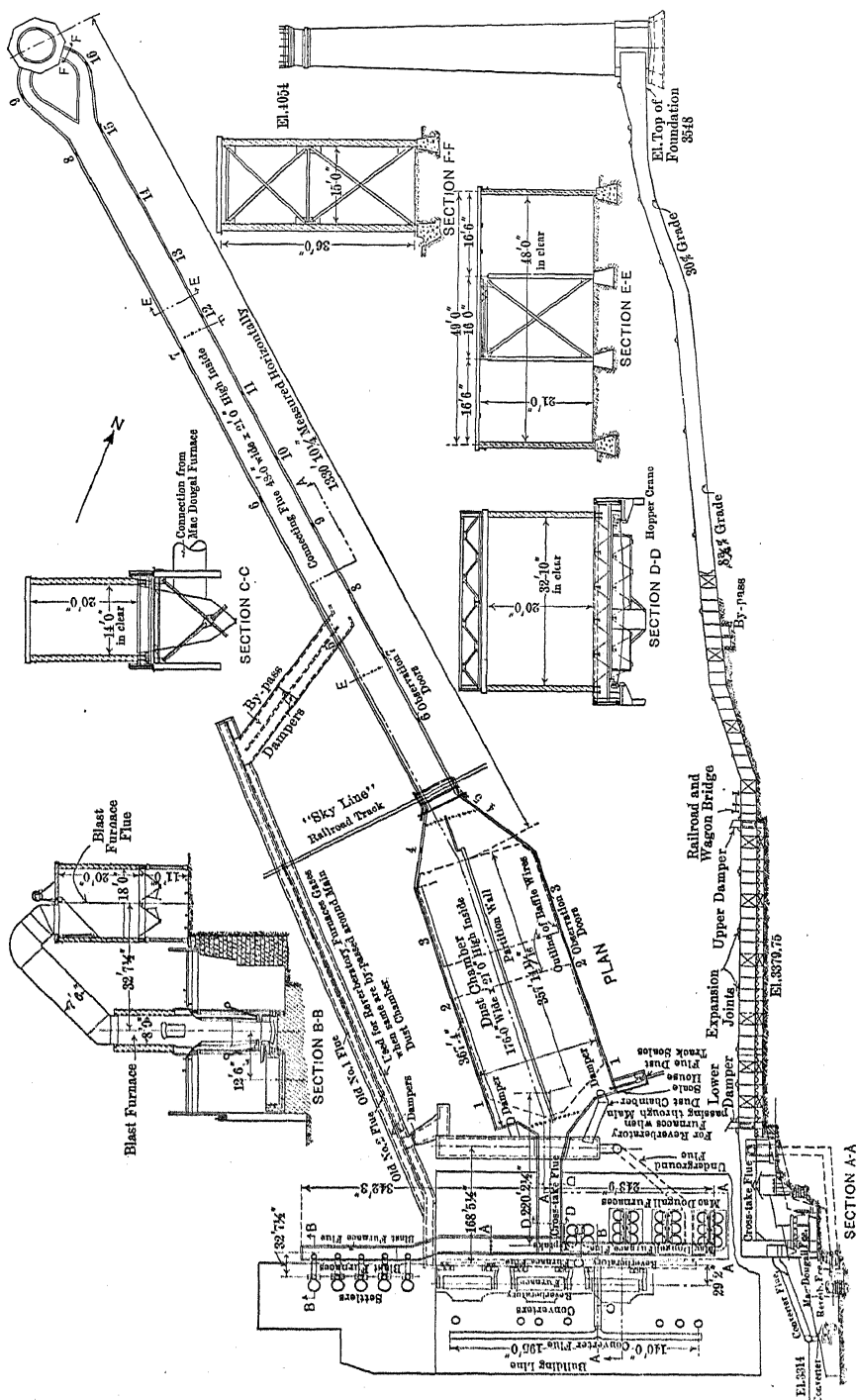
PLATE XIII.—NEW FLUE SYSTEM,

a hinged damper which is closed when the furnace is not in operation.
The tracks under the hoppers in this flue are on the reverberatory-furnace
charge-floor level, so that the regular calcine trammer's crew draws and
trams this dust to the reverberatory furnaces without extra expense for
labor.   A similar arrangement is obtained with the MacDougall furnace
flue, a cross-section of which is also shown.

The uptake, which carries the combined converter, MacDougall and
blast-furnace flue gases, is rectangular in section, being 32 ft. 10 in. long
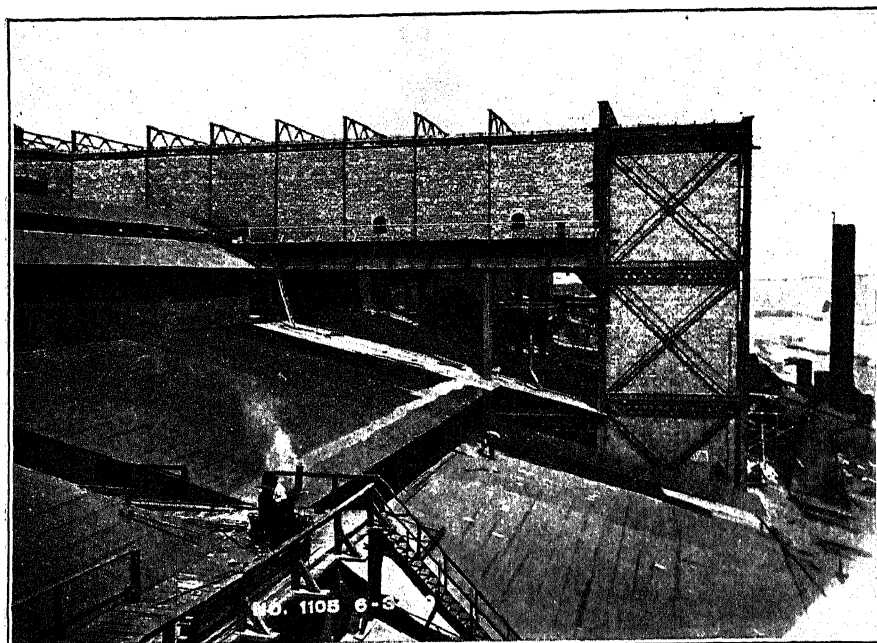
PLATE XIV.—UPTAKE AND CROSSTAKE.

by 20 ft. wide, inside dimensions, by 93 ft. 9 in. high from top of hoppers.
The blast and MacDougall furnace gases enter from opposite sides of
this uptake, the bottom of the MacDougall flue being about 9 ft. above
the top of the blast-furnace flue.   In order to prevent any baffling action
of one upon the other and at the same time to help the two gases to mix
more readily and thus prevent their being carried along their respective
sides of the flue and dust chamber, a steel plate diaphragm was hung in
the uptake, which has the effect of causing a layer of blast-furnace gases
to travel along the bottom of the flue instead of one side, while the
MacDougall furnace gases will travel along the top.

A section D-D of the cross-take flue which connects the uptake with
the main dust chamber is shown in the plate referred to above.   The
entire bottom of this flue is equipped with hoppers, from which the dust
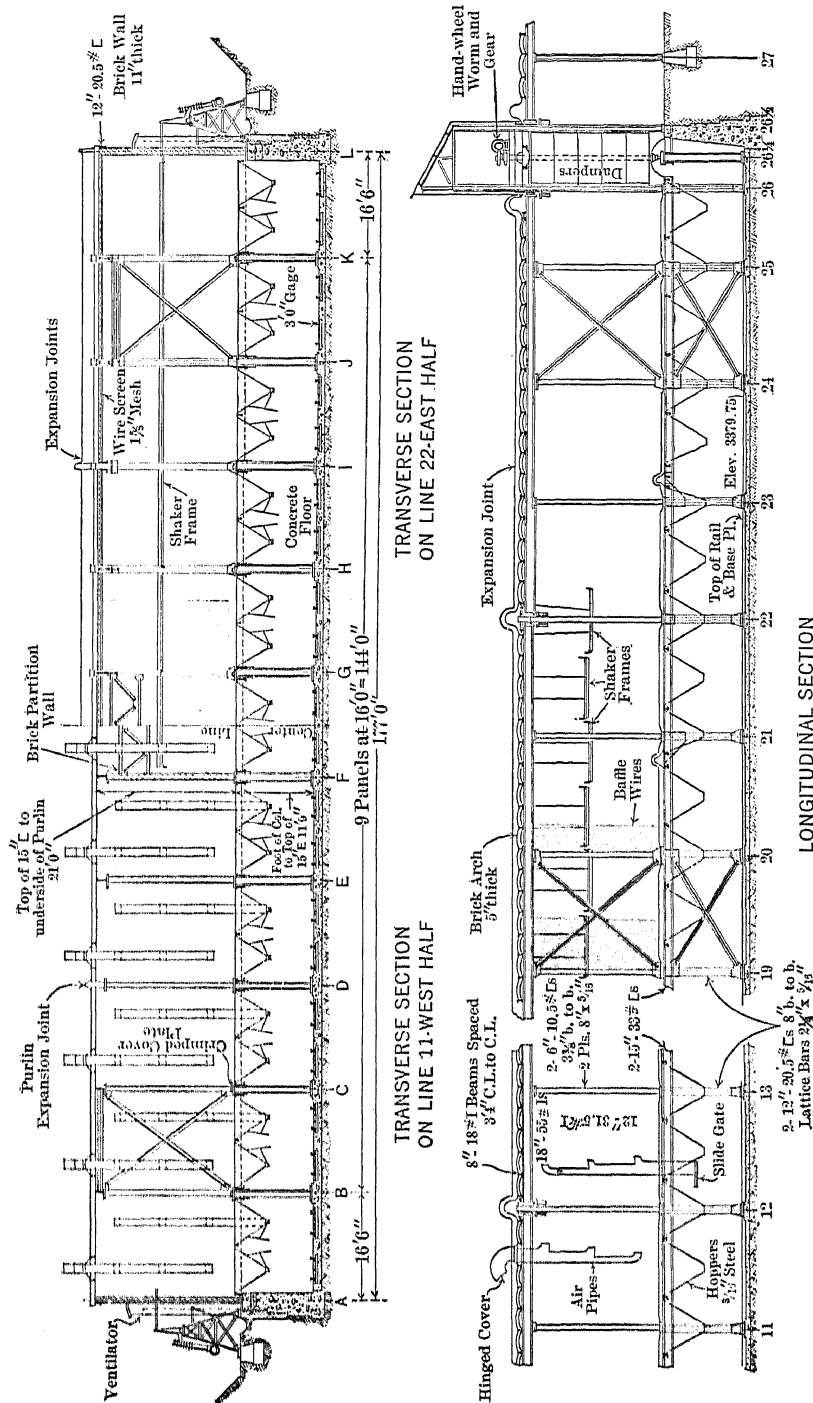
PLATE XV.—MAIN DUST CHAMBER.

PLATE XVI.—HOPPER PLAN OF DUST CHAMBER.

is drawn into hoppers on a crane and is thereby transferred to chutes leading to the bottom of the uptake flue, where it is drawn out and handled in the same manner as described above. Plate XIV gives a general view of the uptake and cross-take.

In the flue connecting the main dust chamber to the chimney, no provision was made for removing the accumulated dust, the bottom being simply a level dirt floor, as shown in section E-E. Openings 3 ft. 3 in. by 5 ft., provided with hinged cast-iron doors, are located about every 110 ft. on the east side and about every 240 ft. on the west side. It will be noted from the general plan that this flue branches near its upper end, so as to enter the chimney at two points 90° apart. The openings in the base of the chimney are each 36 ft. high by 15 ft. wide. From the point where the flue branches, each leg grows narrower and higher as so to maintain a practically uniform sectional area. Section F-F shows a view of one branch, at a point near the chimney entrance.

VI. Main Dust Chamber.

A cross-section and partial longitudinal section of the main dust chamber is shown in Plate XV. These sections are taken near the upper end of the chamber, in order to show special features of construction such as wire-shaking arrangement, cold-air pipes, etc. Other details of construction are the same throughout. Plate XVI shows a hopper plan of the dust chamber. The part within the dotted lines is hung with steel wires spaced at about 2.3 in. center to center, there being a total of about 1,200,000 wires, each of which weighs about 1 lb. As indicated on the plan, these wires are hung in two groups or divisions, there being about 47 ft. of clear flue between them. It will be noted that a clear space is left at the inlet end, affording an unobstructed passage for the gases to distribute themselves equally over the full width of the chamber. The total average length of flue filled with wires is approximately 317 ft. There are two sizes of wire used in the first group or division. For a distance of about 51 ft. from the inlet end, No. 8 W. & M. wires, 16 ft. long are used, the rest being No. 10 wires, 20 ft. long. The second division is hung with No. 10 W. & M. wire 20 ft. long.

At the upper end of the first division of wires, there are 22 pipes for the admission of air in case it should be desired to lower the temperature of the flue gases, with a view to condensing fume.

These pipes are shown on Plate XV and by photograph on Plate XVII, which plate also shows other structural details.

The open space between the two groups of wires is provided in order that a thorough intermingling of the cold air and gases can take place before entering the second division.

It will be noted that 11 of these air pipes enter from the top, or roof, of the flue and 11 through the bottom.

A brick partition wall, on the F line of columns, divides the chamber into two parts, there being six panels on the east and five on the west side. At each end of the chamber a line of butterfly dampers is provided
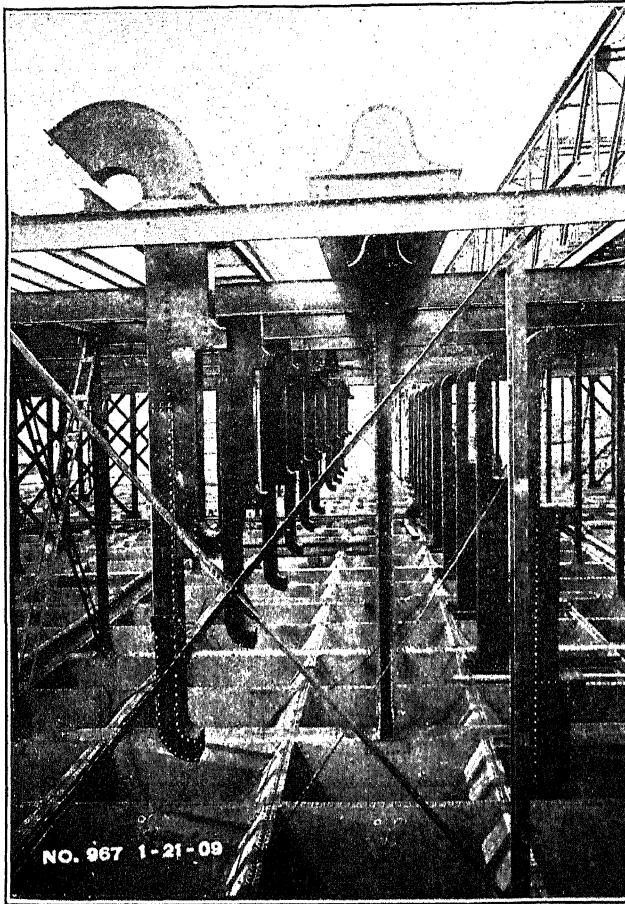
PLATE XVII.—VIEW OF COLD-AIR PIPES, EXPANSION JOINTS, AND HOPPERS.

so that either side, or the whole, of the chamber may be closed to the passage of gases. Plate XVIII shows these dampers partially erected. They are made of heavy cast-iron plates which are carried by a vertical shaft. The step bearing at the bottom is below the floor plate, where it is free from dust and can be oiled and inspected. The upper end of the shaft passes out through the roof of dust chamber, each one being equipped with a worm wheel, worm, and hand wheel for operating the damper.

Guided by the observations made in the experimental flue, which showed that the dust and condensed fume collected on the wires at temperatures about 400° and below, the upper division of wires in the new chamber was equipped with a device for shaking them in order to remove the condensed fume and dust. We estimated that the temperature in the first or lower division of wires would be above the point where the fume would condense and collect on the wires, but in this we were in error. The improved draft conditions provided by the new system resulted in the entrance of so much more cold air at the various furnace
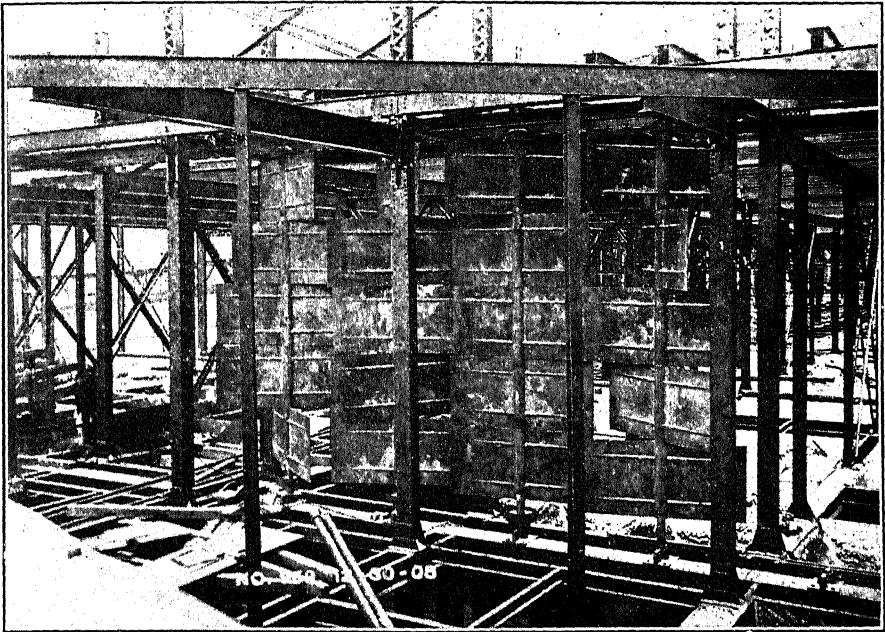
PLATE XVIII.—VIEW OF DAMPERS AT INLET END OF DUST CHAMBER (Partly erected).

and flue openings than under the old system, that a temperature of under 400° F. exists in the first division of wires with the result that there is an accumulation of solid matter upon them. In order to remove this dust, connections were made to the old stone dust chamber which carries the hot reverberatory gases that are normally passed around the new chamber. At intervals the lower dampers on one side at a time of the new chamber are closed and these gases are turned into that side. This results in volatilizing and disintegrating the material on the wires so that it breaks up and the greater part of it falls off.

Plate XIX shows the manner in which the wires are suspended. Crimped iron screen cloth 1⅝-in. mesh is stretched along the bottom of the roof purlins. Flat bars 2.5 in. by 5/16 in., bolted to the purlins,

hold the screen in place. The warp or longitudinal wires are No. 8 W. & M. gauge, while the woof or cross wires are No. 10 W. & M. gauge. The baffle wires are provided with a hook in the form of a shepherd's crook on one end and are suspended from alternate intersecting points on the screen. This arrangement has the effect of staggering the wires which are about 2.25 in. apart, making the baffling action more effective.

The device for shaking the upper division of wires consists of angle-iron
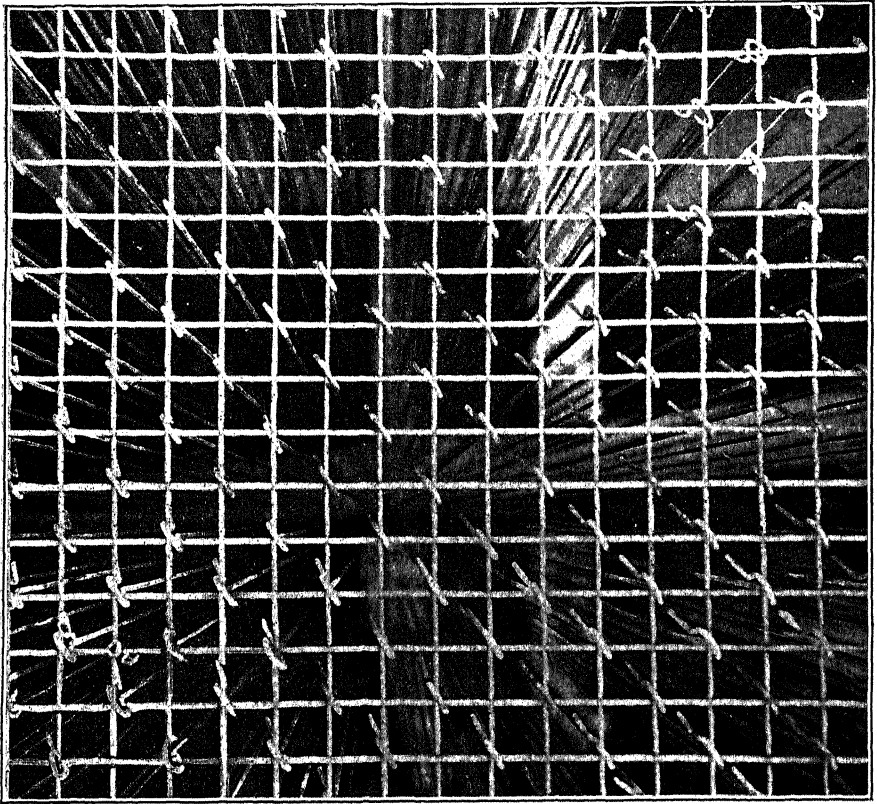
PLATE XIX.—METHOD OF SUPPORTING BAFFLE WIRES.

frames suspended from the roof beams by hangers about 10 ft. in length, thus bringing the frames at about the middle point of the baffle wires. On these frames a wire netting having a 4 by 7 in. mesh is stretched, through which the baffle wires pass. The frames are 10 ft. wide and extend from the side walls to near the partition wall. A connecting rod attached to each frame passes out through the flue wall and is connected to a bell-crank lever which is actuated by an eccentric, giving the frame a stroke of 9.5 in. The frames make about 60 strokes per minute.

Line shafts, carrying the eccentrics on each side of the flue, are operated

by electric motors. The eccentrics are set at different angles in order to prevent vibrations which might be produced in the structure if the complete group of wires was moving in unison.

The wires are shaken for a period of about 30 min. at intervals of from 60 to 90 days as may be required. While the shaking device on one side of the chamber is in operation all gases are passed through the other side. The upper dampers are closed on the side where the wires are being shaken, thus guarding against the loss of any dust removed from the wires.

Plates XX and XXI show the wire-shaking device during construction.
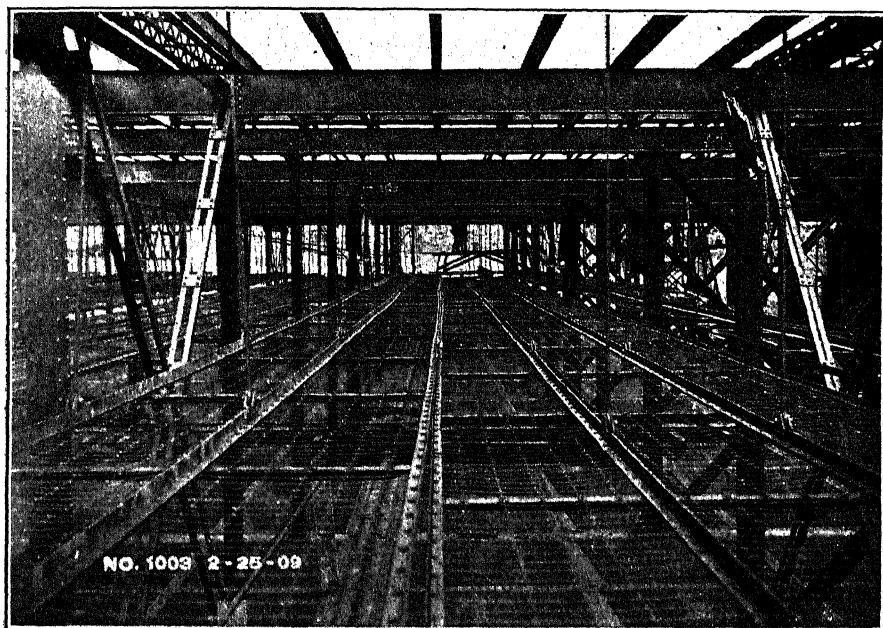
PLATE XX.—DEVICE FOR SHAKING WIRES. (Baffle wires not in place.)

VII. SOME DETAILS OF CONSTRUCTION.

Throughout the flue system a structural steel framework is used. The inner columns are of box section, being built up from plates and channels, while the columns in the outside wall are 12-in. I-beams, between which a brick curtain wall 11 in. thick is laid. The main roof beams resting on top of columns are chiefly I-beams, channels being used only at the expansion joints.

The roof consists of steel I-beam purlins between which brick arches 5 in. thick are sprung. Special perforated bricks were made for this purpose, the skewback brick being molded to fit the I-beams, while all others were made radial to suit the arches. The valleys between the skewback brick and top of I-beams were filled with concrete (see Fig.

1, Plate XXII) and, when the roof was completed, it was given two coats of a one-to-one concrete wash, in order to fill up all cracks and prevent leakage of air or water.

For the walls of the flue, which are 11 in. thick, special perforated brick were also made. These brick were 11 in. long by 5⅜ in. wide by 4⅜ in. thick. They are laid up to break joints, one row of headers to
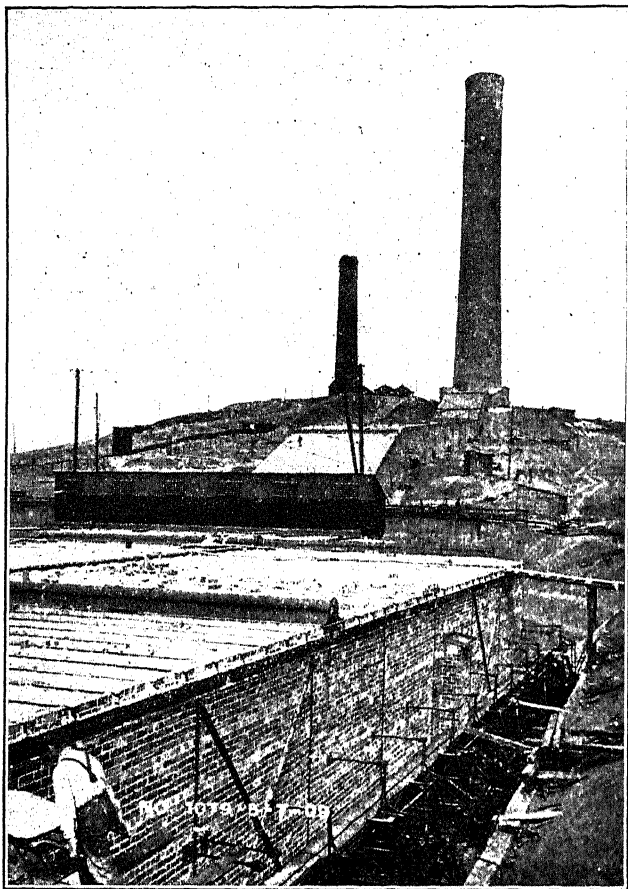
PLATE XXI.—WIRE-SHAKING MACHINERY AND GENERAL VIEW OF FLUE SYSTEM.

four of stretchers, and were all made at the plant which was constructed for the manufacture of brick for the chimney.

Each column in the dust chamber, for a distance of about 18) ft. from the inlet end, was provided with a hole in the base and another near the top, so that it could be cooled by allowing cold air to blow through it into the flue, in case the temperature should become high enough to endanger its strength. We have never found it necessary to use these openings.
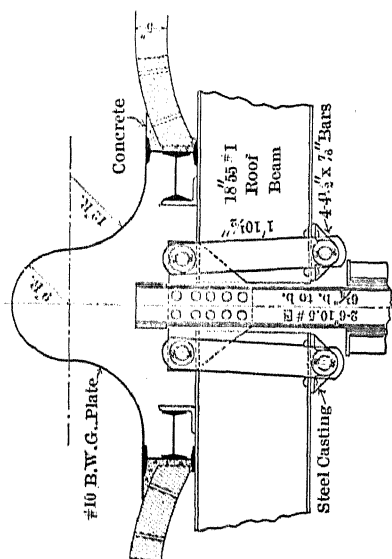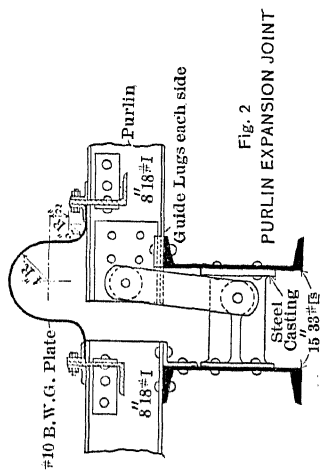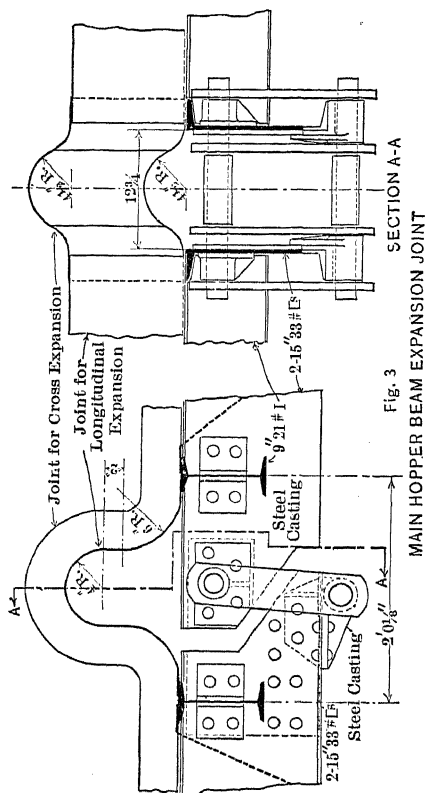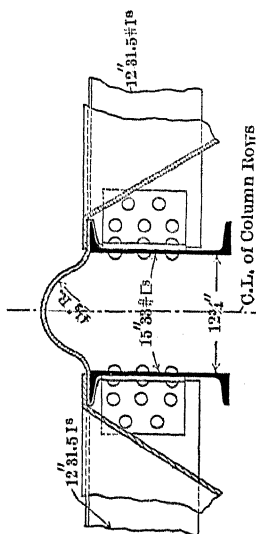
PLATE XXII.—DETAILS OF EXPANSION JOINTS.

In a structure of this size and character, where a goodly part of the steel is exposed to high and varying temperatures, we considered it very essential to the life and maintenance of the structure to make ample provision for the expansion and contraction of the steel. We were also advised by our friends in Anaconda, who had experienced some misfortunes with similar structures, to make all parts of good heavy construction. We believe that it is to the exercise of these precautions that we

SECTION ON LINE E-F   SECTION ON LINE C-D

SIDE ELEVATION
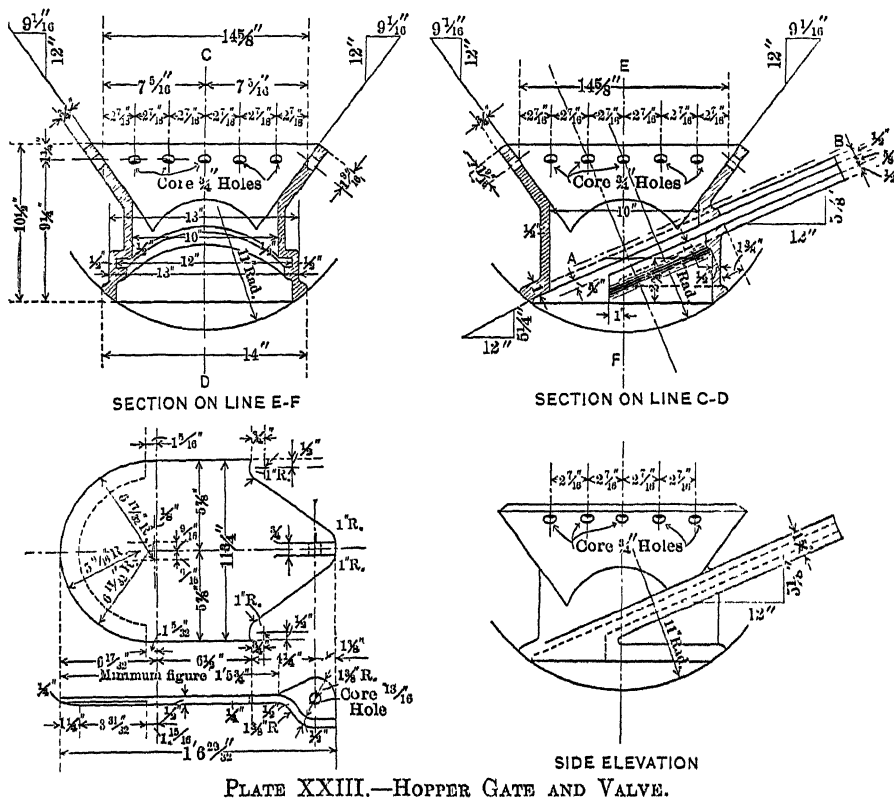
PLATE XXIII.—HOPPER GATE AND VALVE.

are able to say at this time, after the flue has seen four years of continuous service, that it stands as true and plumb as upon the day it was erected.

To provide for this expansion the framework is built in units of from three to seven panels each. The middle panel is securely braced, which holds it plumb, the expansion taking place in both directions from this central point. The roof beams are riveted to the tops of all columns except the one midway between the braced panels, where they are suspended from the columns by links as shown in Plate XXII, Fig. 1. The roof between the purlins on each side of this expansion joint is made of No. 10 steel, bent as shown in the figure. These joints extend across the full width of the flue.

To take up the expansion in the purlins which extend across the flue two expansion joints are provided running the full length of the dust chamber, thus dividing the roof into three sections as indicated in the sectional elevation Plate XV. The middle panel in each of these sections is securely braced, so that the expansion is taken up at the joint, a detail of which is shown in Fig. 2, Plate XXII.

The main-hopper beams are 15-in. channels, which are riveted to the sides of the lower columns. A detail of the expansion joints in these beams is shown in Fig. 3, Plate XXII.

In the longitudinal section shown in Plate XV it will be seen that
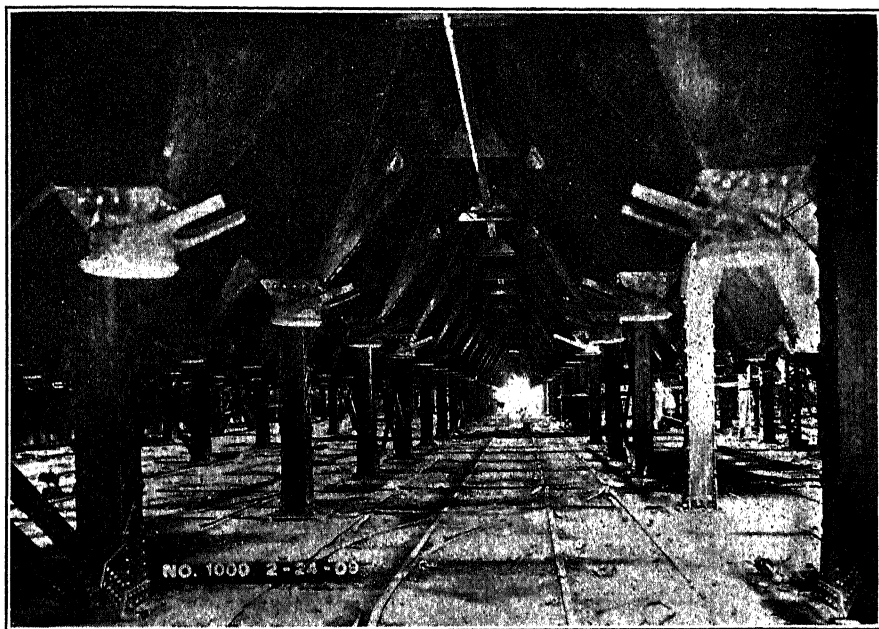
PLATE XXIV.—HOPPERS UNDER DUST CHAMBER.

there are two hoppers to each longitudinal panel, the center line of the hoppers coinciding with the center line of the panel and of the columns.

The object of this arrangement is to allow the expansion of the cross-hopper beams to be taken up by the spring in the main longitudinal beams. A crimped plate covers the space between the main channel beams as shown in Fig. 4, Plate XXII.

A part of the structure which might seem of minor importance, but which is operated more frequently than any other single element, is that of the gate on the bottom of the hoppers. When it is considered that there are 1,042 hoppers under the main dust chamber, 100 under the cross-take flue, 74 under the blast-furnace flue, 12 under the MacDougall flue
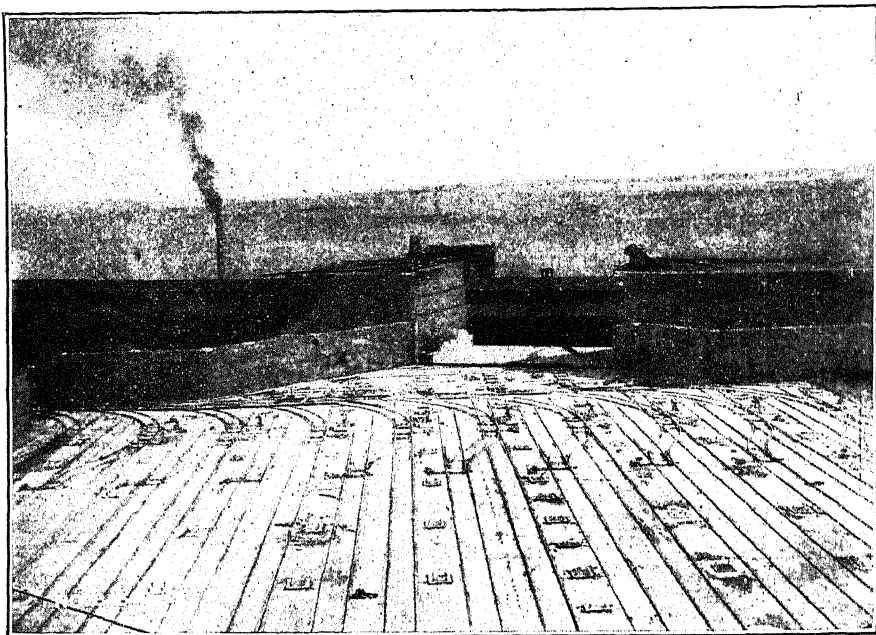
PLATE XXV.—TRACKAGE UNDER DUST CHAMBER.
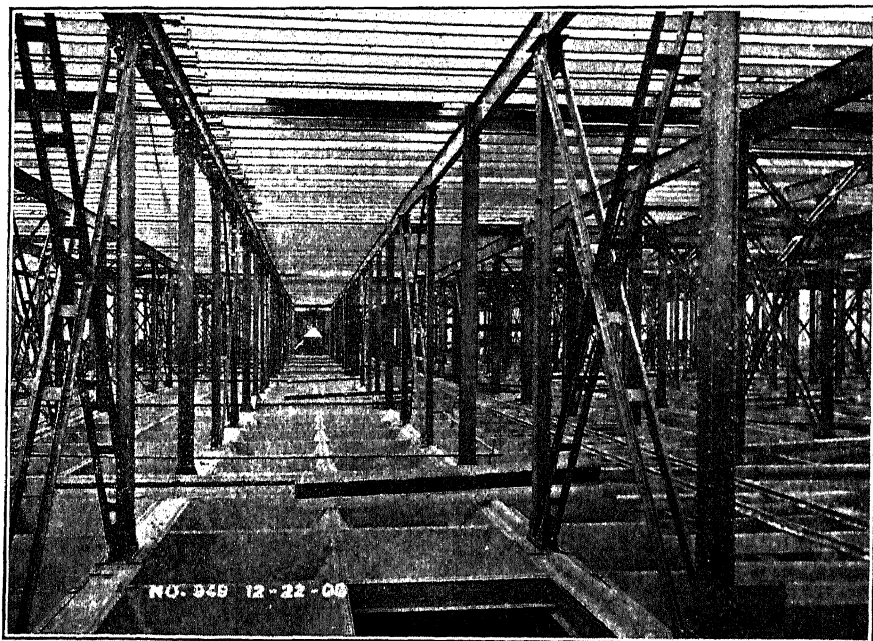
PLATE XXVI.—INTERIOR OF DUST CHAMBER DURING ERECTION.

and eight under the uptake, some of which are being operated every day, the desirability of an efficient and easily operated hopper gate is appreciated. Plate XXIII shows the gate which was designed for this work. It has given extremely good satisfaction. It is made of cast iron, all holes being cored so that it required no machine work of any kind. The essential feature of this gate is that there are no closed slots to fill up with dust and cause the slide to stick or to prevent it from closing.

The hoppers and tracks under the dust chamber are shown in Plate
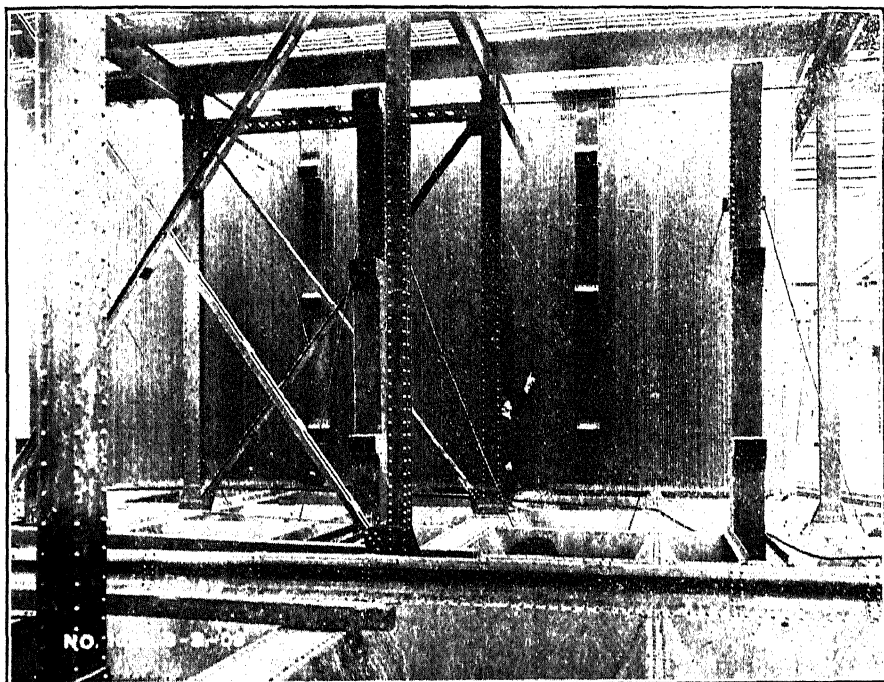
PLATE XXVII.—INTERIOR OF DUST CHAMBER, SHOWING WIRES AND COLD-AIR PIPES.

XXIV, and Plate XXV shows a portion of the track system before any steel was erected. The total length of trackage under the dust chamber is about 8,900 ft.

Dust is transferred from the main dust chamber to the furnaces in hopper-bottom cars of about 80 cu. ft. capacity. The over-all length of each car is just equal to the span on one hopper, so that one spotting only of the train is necessary for each loading.

Some additional details of dust-chamber construction are shown on Plates XXVI and XXVII.

VIII. THE CHIMNEY.[1]

Having decided upon the construction of a chimney 506 ft. high and 50 ft. inside diameter at the top, the contract for furnishing the brick and building the chimney was let to the Alphons Custodis Chimney Construction Co., of New York, who designed and erected the structure. The foundation was put in by the owners, the design and construction having the approval of the chimney contractor, who is responsible for the stability of the whole structure.

The chimney is designed to withstand a wind velocity equivalent to 33 1/3 lb. per square foot diametral projection. The total estimated weight of brickwork in the completed chimney is 17,000 tons. The thickness of the shell is so proportioned that the combined dead and wind load do not exceed 21 tons per square foot at any point. There is no tension at any point on the windward side. Specifications for the brick of which the chimney is constructed required that they withstand a test pressure of not less than 3,750 lb. per square inch of total sectional area, including perforations. General drawings of the chimney and of some details of construction are shown on Plate XXVIII, while Plate XXIX. is a view of the completed structure.

The whole of the interior is circular in section. The exterior is octagonal for a height of 46 ft. above the base, the remainder being circular.

The profile of the chimney presents four separate tapers as follows: The upper 180 ft. is 2 per cent. taper (1 per cent. batter); the next lower 100 ft. is 4 per cent. taper (2 per cent. batter); the next lower 180 ft. is 7 per cent. taper (3.5 per cent. batter); and the octagonal base 46 ft. in height is 8 per cent. taper (4 per cent. batter). The wall thickness varies by steps of 2 in., the sections of constant thickness being of variable heights, ranging from 10 ft. at the bottom of the circular section, where the wall is 54 in. thick, exclusive of lining, to 50 ft. at the top, where the wall is 18⅛ in. thick.

The chimney is lined throughout with a 4-in. course of hard-burned brick laid in acidproof mortar, the joints being made as thin as possible. This lining is what is known as sectional construction, each section resting on a corbel ring built out from the inner wall of the main chimney shell. No section of lining exceeds 20 ft. in height. An air space of 2 in. is left between lining and main chimney shell. Fig. 1, Plate XXVIII, shows a detail of corbel and lining. The top brick of the corbel is molded with a lip for the purpose of forming a drip outside of the lining. In order to

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[1] New 506-ft. Chimney at Great Falls Smelter, *Engineering and Mining Journal*, vol. LXXXVII, p. 156. The World's Largest Chimney, *Engineering News*, vol. LX, p. 583. The Chimney of the B. & M. C. C. & S. M. Co., *Engineering Record*, vol. LVIII, p. 600. The World's Largest Chimney, *Mines and Minerals*, vol. XXX, p. 257.
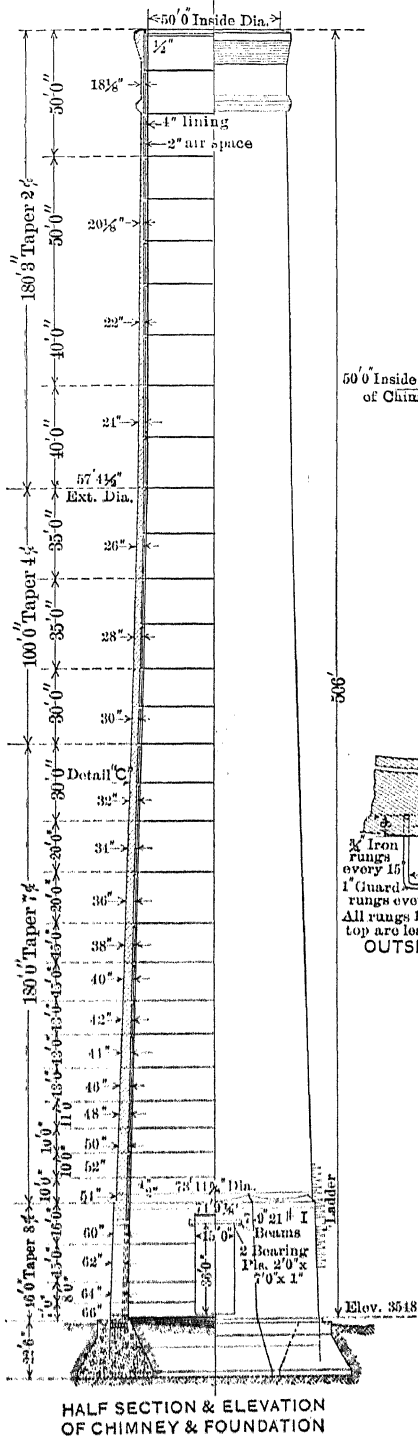
PLATE XXVIII.—DETAILS OF 506-FT. CHIMNEY.
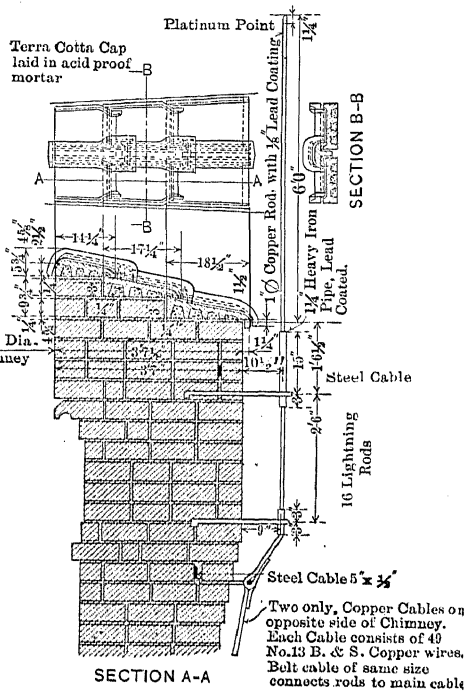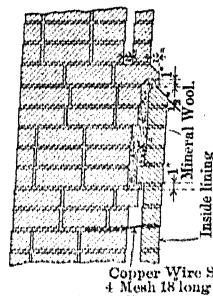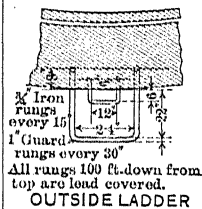
prevent any dust or condensed fume from settling in the space between lining and chimney wall, this space was filled with mineral wool for a depth of about 2 ft. at the top of each section. The acidproof mortar, which is composed of silicate of soda, asbestos wool and other ingredients, is a very slow-setting material and, when fresh, acts as somewhat of a
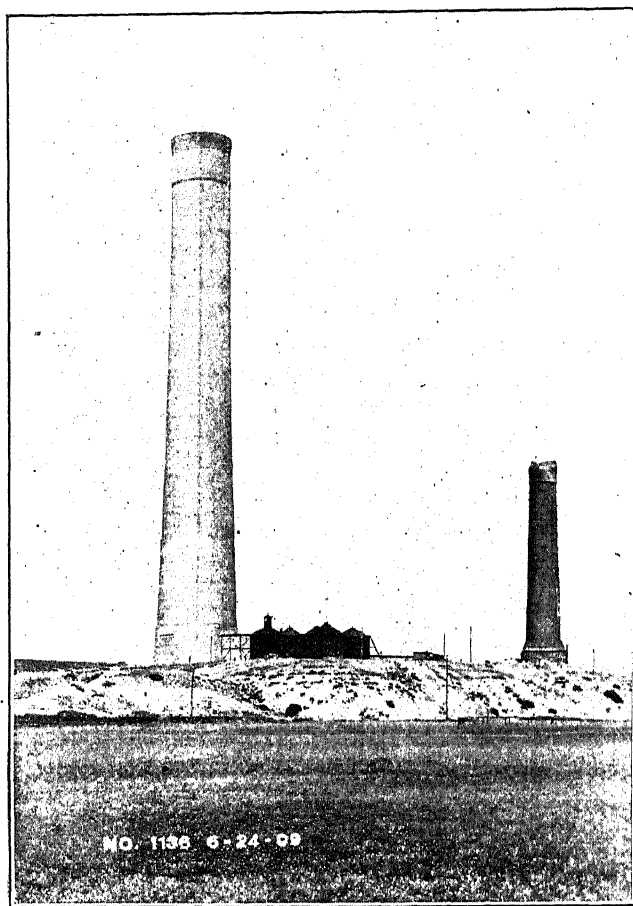
PLATE XXIX.—GREAT FALLS CHIMNEY, 506 FT. HIGH.

lubricant between adjacent courses of brick. It is, however, extremely hard when thoroughly set and dry. In order to hold the lining course in place until the mortar had set, small sections of copper wire netting were built into the main shell and lining at intervals of 7 or 8 ft.

The mortar used in the main part of the chimney was a 1:2:5 cement-lime-sand mixture. For a distance of 50 ft. down from the top, all outside joints in brickwork were pointed up with acidproof mortar and the top 3 ft. of the main wall was laid with this mortar throughout.

The top of the chimney is corbeled out in steps of 0.75 in. to a thickness of 42 in. The cap is composed of specially shaped radial interlocking terra-cotta tiles laid in acidproof mortar. Fig. 2, Plate XXVIII, shows the general design and arrangement of this cap.

For protection against lightning, 16 rods are provided, each of which extends 5 ft. above the chimney top. They are made of 1-in. round copper rod, lead coated for protection against acid, and tipped with a platinum point 1.25 in. high. These rods are all connected to a copper cable which encircles the chimney a few feet below the top. From this encircling cable two ⅝-in. copper cables lead to the ground on opposite sides of the chimney. The lower end of each cable is fastened to a copper plate about 6 sq. ft. in area, which is buried several feet under ground and at a distance from the foundation, at a point where moisture is usually present. The cable band, as well as the vertical cables for a distance of about 100 ft. down from the top, are lead covered. The cables are composed of 49 No. 13 B. & S. gauge copper wires. Fig. 2, Plate XXVIII, shows the general arrangement and method of supporting the lightning rods.

An outside ladder built into the brickwork extends from the base to the top. It consists of one-piece rungs made of 0.75-in. round iron, which stand out 6 in. clear from the chimney wall. The hooked ends of these rungs extend 6 in. into the chimney wall. They are placed at every third joint, or about 15 in. apart. At every second rung there is a guard rung or loop made of 1-in. round iron. The hooked ends extend 8 in. into the brickwork, and the loop is 16 in. outside of the ladder rung and is 28 in. wide inside. For a distance of 100 ft. down from the top, the ladder and guard rungs are lead covered.

At each corbel point on the circular portion of the chimney, a flat steel cable about 5 in. wide by 0.5 in. thick is laid in the brickwork. These are for the purpose of reinforcing the shell at the corbel points, where a small portion of it is exposed to the heat of the gases. The cables used were old hoisting cables that had been discarded at the mines.

There are four openings in the base of the chimney, each of which is 36 ft. high by 15 ft. wide. The brickwork across the top of each of these openings is carried by seven 9-in. 21-lb. I-beams, which rest on 1-in. plates at each end, and are protected from the acid fumes by special brick fitted over the flanges. Only two of these openings are in use at the present time, the other two being bricked up with double curtain walls which are flush with the inside and outside of the chimney walls. The outside, or facing, course of brick in all flue openings is laid in acid-proof mortar, and a course of brick 1 ft. thick, laid in the same material, covers the floor of the flue openings and protects the concrete base at this point. Plate XXX is an interior view at the base of the chimney,
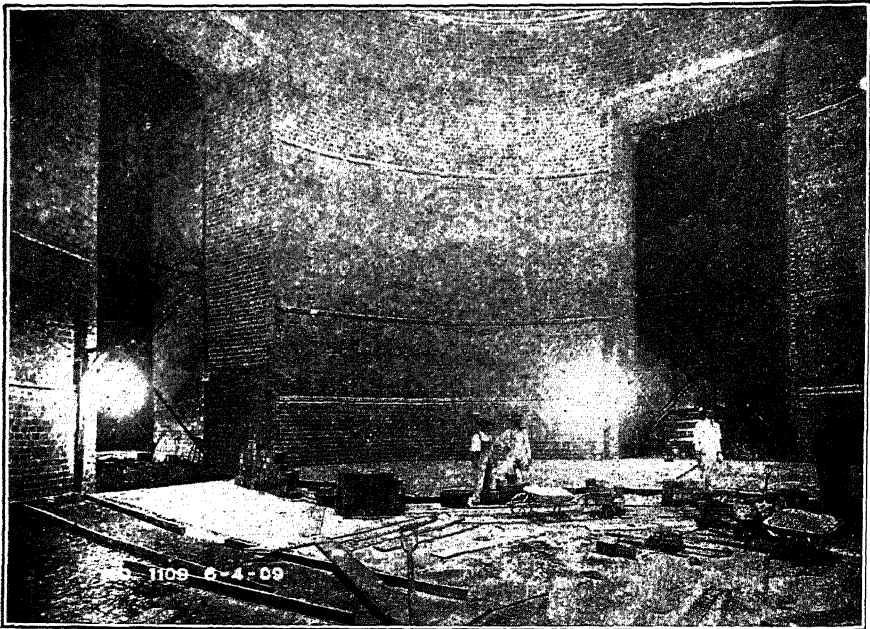
PLATE XXX.—CHIMNEY BASE INTERIOR.
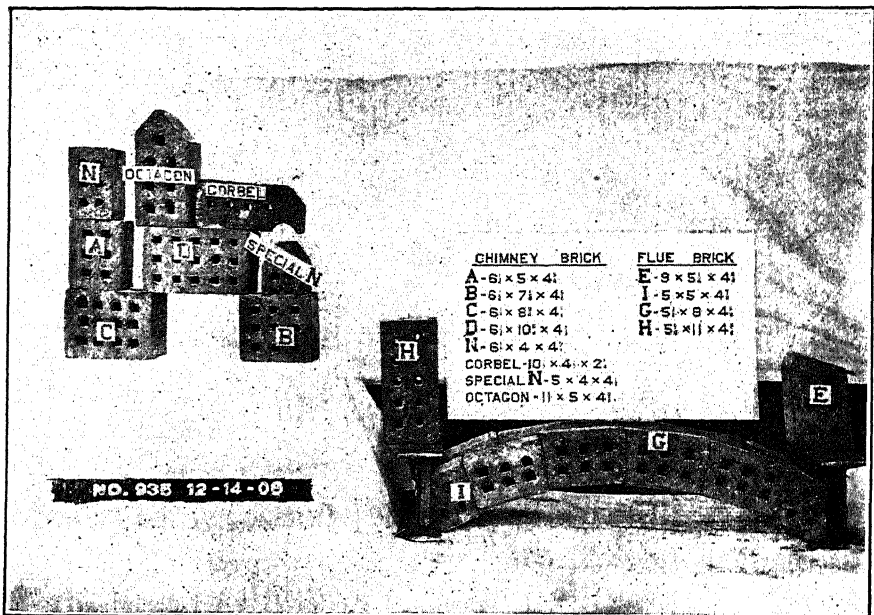
PLATE XXXI.—BRICK USED IN CHIMNEY AND FLUES.

showing the flue entrances. It also shows several corbel rings and a portion of the paved floor which is described later.

In order to break joints and also to obtain the required variations in the thickness of the walls, five sizes of brick were used in the main chimney shell. These are known as perforated radial brick. The holes are about 1 in. square and are in a vertical position when the brick are laid. Plate XXXI shows the various sizes and shapes of brick used in the construction of the chimney and flues.

To supply these brick, a plant was erected near the chimney site, equipped with the necessary grinding pans, etc., an auger machine of over 100 tons capacity in 8 hr., a drier which is arranged to use either direct or waste heat, and eight circular beehive kilns, independently fired. Altogether, the plant has a capacity of 100 tons per day and is being operated at the present time for the manufacture of such products as paving brick, fire brick, hollow tile, drain tile, etc., as well as common and perforated building brick. The shale of which the brick was made was quarried with steam shovel, near the plant. An analysis of this shale is as follows:

Loss on Ignition	Insoluble	SiO_2	Al_2O_3	FeO	CaO	MgO	SO_3
8 02	77 8	58 3	19 0	6.3	1.2	2 3	0.74

Crushing tests were made on several samples of brick, with results as shown in the following table. In all cases pressure was applied parallel to the perforations. In calculating the crushing weights, the gross area of the brick was used, no deduction being made for the area of holes.

	PRESSURE IN LB. PER SQ. IN.	
	First Crack	Ultimate
Hard burned......	3,033	7,173
Medium burned..	3,064	4,255
Soft burned........	2,766	3,949

Chimney Foundation.

The chimney foundation is shown on Plate XXXII. It is composed of concrete, and is annular in shape, the inside being circular, and the outside octagonal.

It contains 4,300 cu. yd. of concrete composed of a 1:2.3:4.5 mixture of cement, tailing sand from the concentrator, and broken slag from the smelter. A drainage gutter near the top having a slope of ⅛ in. per foot

SECTION A-A

SECTION B-B
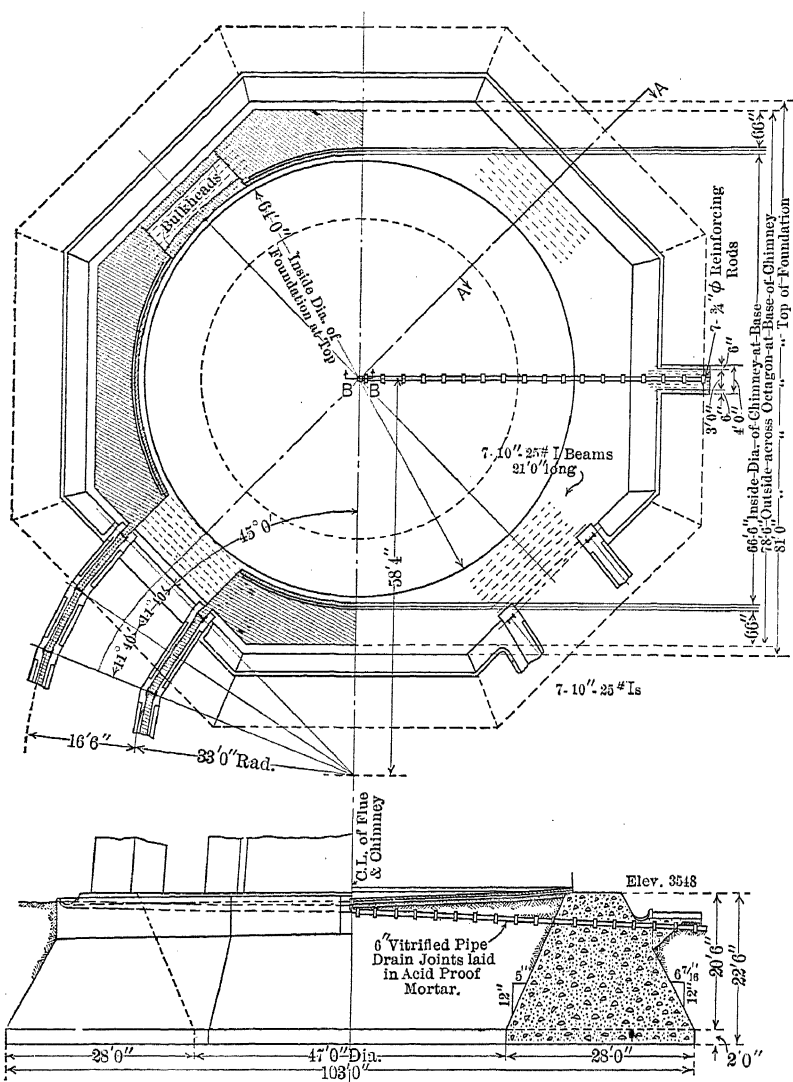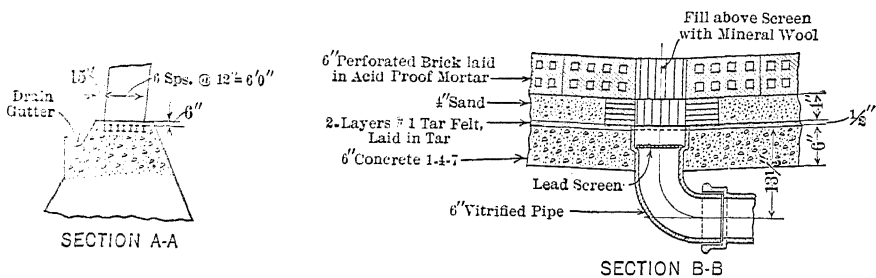
PLATE XXXII.—FOUNDATION FOR 506-FT. CHIMNEY.

was built as a part of the foundation. The footing on which the founda-
tion rests is a hard shale. A loading test was made on the bottom of
the excavation to determine whether it would withstand the required
maximum pressure which it was estimated would be 5.25 tons per square
foot. Four cast-iron plates 2 ft. square by 3 in. thick were set in a thin

Flue System—Construction Data.

EXCAVATION:

	Cu. Yd.
Main dust chamber	90,354
Connecting flue	14,800
Chimney	5,500
Track to smelter	10,000
Total	120,654

CONCRETE:

Chimney	4,300
Flue system.	7,933
Total	12,233

STEEL AND IRON WORK:

	Tons
Structural steel	3,412
Wire baffles (1,200,000).	600
Wire netting (89,400 sq. ft.)	35
Castings	203
Total	4,250

BRICKWORK:

	Cu. Ft.	Tons	Equivalent in Common Brick
Blast-furnace dust chamber	15,968	926	351,000
MacDougall dust chamber	6,505	377	143,000
Uptake	13,100	760	288,000
Cross-take.	7,977	463	176,000
Main dust chamber	63,356	3,675	1,394,000
Connecting flue	91,866	5,328	2,021,000

	Cu. Ft.			
Chimney		293,104	17,000	6,441,000
Shell	261,429			
Lining	29,948			
Floor	1,727			
Totals		491,876	28,529	10,814,000

TRACKAGE:

	Ft.
Under dust chamber (40-lb. rail)	8,900
Dust chamber to smelter (55-lb. rail)	4,300
Under blast furnace flue (40-lb. rail)	700
Total	13,900

cement grout and at equal elevations. These plates were at the corners
of a rectangle 6 ft. 6 in. by 16 ft. 6 in. Plate girders were then sym-
metrically placed on these plates, and the whole loaded with rails sufficient
to produce a load 6.5 tons per square foot of bearing area of the cast-iron
plates, being about 24 per cent. in excess of the maximum calculated pres-
sure. The test was continued for a period of 20 days, during which time

heavy rains flooded the excavation to a depth of several inches over the bearing plates, making the test an especially severe one. The total settlement of the bearing plates was 0.058, 0.058, 0.075, and 0.088 ft. respectively, from which it was decided that the footing material was safe for the required loading.

To prevent possible damage to the concrete foundation by the action of leachings from dust on the floor or sides of the chimney, in case the plant should be shut down for an extended period, the floor was given a slope of 0.5 in. per foot to the center, from which a vitrified pipe laid in acidproof cement leads to the outside of the foundation. The floor itself is constructed from the bottom up as follows: 6 in. of concrete footing; two layers of No. 1 felt with a good coating of tar under, between, and on top of same; 4 in. of sand to act as a bed for brick flooring and also to prevent the heat of gases from damaging the tar and felt; and 6 in. of good quality perforated brick laid in acidproof mortar. The details of this construction are shown on Plate XXXII.

IX.—OPERATION OF NEW FLUE SYSTEM.

The new flue system was put into operation on June 12, 1909. It has been in service continuously ever since, with the exception of a period of about 40 days in December, 1909, and January, 1910, when the plant was shut down on account of a strike among employees of the railway company, which stopped the delivery of ore and other necessary material.

From June 12, 1909, to Jan. 1, 1913, the total amount of dust collected in the various parts of the system, as well as the average analysis of the dust covering the whole period of 41 months, is shown in Table VIII.

TABLE VIII.—*Quantity and Analyses of Flue Dust.*

NAME OF FLUE	Tons Dust Total for 41 Mo.	Tons Dust Aver- age per Mo.	Cu Per Cent.	Ag Oz. per Ton	Au Oz. per Ton	Insol.	SiO$_2$	FeO	Al$_2$O$_3$	CaO	S
Blast furnace	87,020	2,122	8.08	2.7	0 019	34 7	26 8	33 0	7 1	1.7	16.1
MacDougall furnace	18,741	457	10.22	3 6	0 023	37 6	29.0	27.1	7.5	0.3	21.3
Uptake and crosstake	17,360	423	12 59	4 1	0.026	34 8	26 0	27 2	7.2	0 7	19 5
Main dust chamber	64,048	1,562	8.61	3 3	0 020	33 3	23 6	14 5	8.0	0 7	11 8
Connecting flue[a]	4,000	98	3 09	3.1	0 012	12.6	8 5	5 4	4.0	0 1	10 6
Total....	191,169	4,662	.	.	......	... .	....	..	...	... ..	..

a Weight estimated. Average analysis is from sample taken in June, 1912, at different points from dust chamber to chimney.

A comparison of the dust recoveries from the old and new flue systems, with relation to the amount of original cupreous material treated, may be of interest, and is shown in Table IX.

TABLE IX.—*Comparison of Dust Recoveries from Old and New Flue Systems.*

	OLD FLUE SYSTEM	NEW FLUE SYSTEM
	Oct. 25, 1903, to June 12, 1909, 68 Months	June 12, 1909, to Jan. 1, 1913, 41 Months[a]
Tons new cupreous material treated at MacDougall furnaces	1,213,890	579,545
Tons new cupreous material treated at blast furnaces ...	1,663,498	898,481
Total tons new cupreous material treated at MacDougall and blast furnaces	2,877,388	1,478,026
Tons dust recovered from blast furnace dust chamber....	50,604	87,020
Tons flue dust recovered from entire system	104,239	191,169
Dust recovered from blast furnace dust chamber in per cent. of total new cupreous material to blast furnaces..	3.04	9.69
Total dust recovered from entire system in per cent. of total new cupreous material treated........	3.62	12.93

[a] Plant not operating during December, 1909, and part of January, 1910.

The experimental brick-flue tests indicated that 3.7 per cent. of the new cupreous material charged to the MacDougall roasters was being lost through the old chimney and flue system. Also, that 2.57 per cent. of the new cupreous material charged to blast furnaces was lost in the same way. Applying these percentages to the respective tonnages of new cupreous material treated by the two departments, when operating with the old flue system, and adding to these results the tonnage of dust recovered, we find that 6.66 per cent. of the new cupreous material charged to the furnaces was carried into the flues as dust. Assuming that at present a practically complete recovery of all dust is made, it will be seen that with the new conditions the amount of dust produced is about twice as much as under the old system.

There are several reasons why this is so, two of which are well defined: First, as will be shown later, there is a greatly increased volume of gas passing through the new system per ton of material treated. This, of course, is the result of a much stronger draft, together with improvements in flue construction which have largely eliminated draft hindrances. In this connection it may be of interest to note that the top of the new chimney is 751 ft. above the blast-furnace charging floor, while the top of the old chimney was only 419.5 ft. above the same floor. Second, since the new system has been in operation, a considerable tonnage of fine concentrate has been charged to the blast furnaces, a practice which was not in vogue under the old system.

The curve on Plate XXXIII may be designated as a cumulative plot of the dust collected and drawn from the wire-baffled dust chamber.
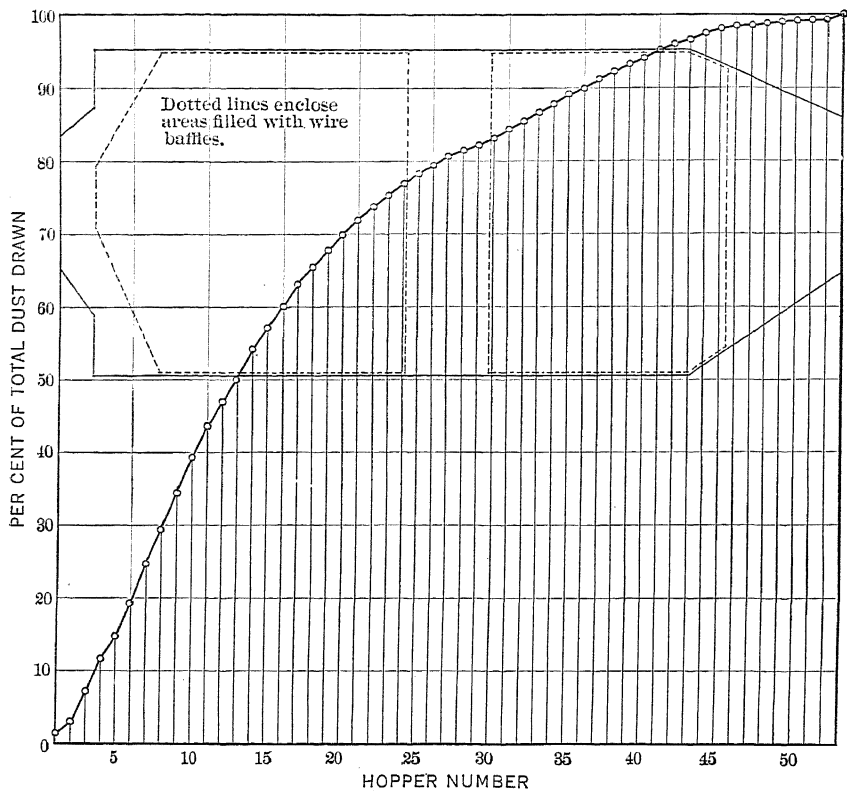
PLATE XXXIII.—CUMULATIVE PLOT OF DUST DRAWN FROM DUST CHAMBER.
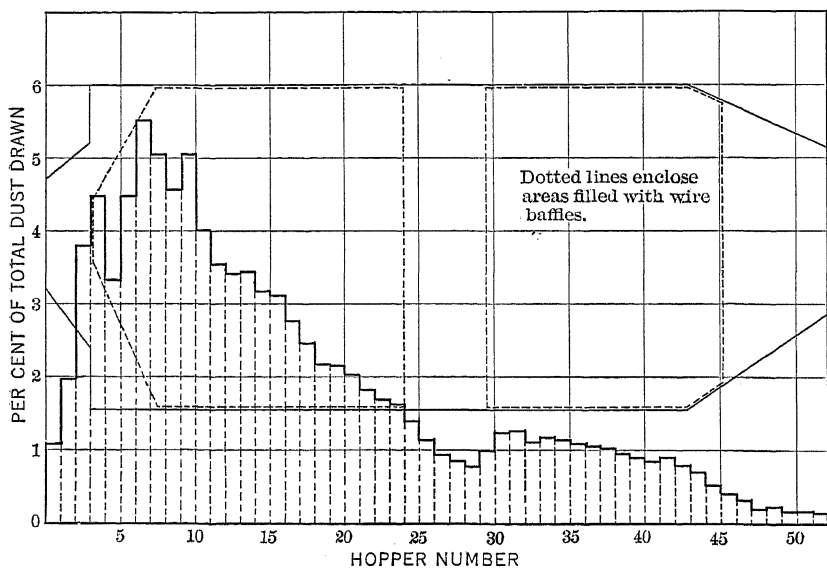
PLATE XXXIV.—PLOT SHOWING RELATIVE DEPOSITION OF DUST THROUGH DUST CHAMBER.

That is, it shows the dust collected from the inlet end up to any given point, in per cent. of the total dust collected in the chamber. The superimposed outline sketch of the dust chamber, showing the wire-baffled areas within the dotted lines, makes clear the cause of changes in the slope of the curve at various points.

The curve or plot on Plate XXXIV shows the relative actual total amounts of dust drawn from the different cross rows of hoppers in the dust chamber. The outline sketch of the dust chamber is again superimposed on this plate to show the location of the maximum dust settlement.

From these two plates it will be noted that about 77.5 per cent. of the total dust deposited is obtained before reaching the upper end of the first section of wires. On account of the low percentage of copper in the dust from the upper part of the chamber, it is evident that the proportion of values collected in the first section of wires is a greater percentage of the total than is indicated by the amount of dust.

Table X shows how the analysis of the dust changes from the inlet end of the dust chamber up to the base of the chimney.

TABLE X.—*Analyses of Dust from Inlet End to Base of Chimney.*

Dust Chamber

	Cu	Insol.	SiO_2	FeO	Al_2O_3	CaO	S
Hopper Nos.							
1–7...	10.05	32 8	23.7	17.4	7.8	0 3	14.3
8–12..	9.17	32 1	22.6	15 2	7.7	0.4	13.1
13–17..	8 72	31 6	23.35	17 2	7.5	1.1	13 1
18–24....	7.43	28.15	20.6	13.45	7.2	1 1	11.7
25–29.	6.83	27 1	19.3	12.0	7.4	1.2	11.2
30–38.	5.83	22 7	15.7	9 7	6.5	1.1	10 9
39–45..	5.17	21.0	14.4	8.7	6.7	1.0	10 7
46–52.. .	3.73	16.3	10.85	6.5	4.6	1.1	10 3

Connecting Flue.

	Cu	Insol.	SiO_2	FeO	Al_2O_3	CaO	S
Door No.							
7–E	4.13	17.4	11.5	7.2	5.6	0.1	10.7
9–E.	3.13	12.9	8.8	5.4	3.6	0 1	10 6
10–E	3.64	15 0	10.5	6.4	4.1	0.1	10.3
12–E..	2 36	9.2	6 1	4.1	3 8	0 1	10.3
Base of chimney....	2.21	8.4	5.4	3.8	2 8	0.1	10.7

Plate XXXV shows the relative deposition of dust across the dust chamber. In making this plot the total weight of dust from hoppers 4 to 43 inclusive was taken, since all other hoppers are in converging portions of the chamber. The view is taken facing the chimney and

shows that a fairly uniform rate of settlement is effected over the full width of the chamber. The marked irregularity in adjacent rows of hoppers in this as well as in Plate XXXIV is due chiefly to the manner in which the dust is drawn. The arrangement of tracks under the hoppers is such as would lead to a result of this kind unless strict supervision is exercised over the operators.

Plate XXXVI shows the distribution of dust collected in the blast-furnace flue, with relation to the furnace locations and furnace days in
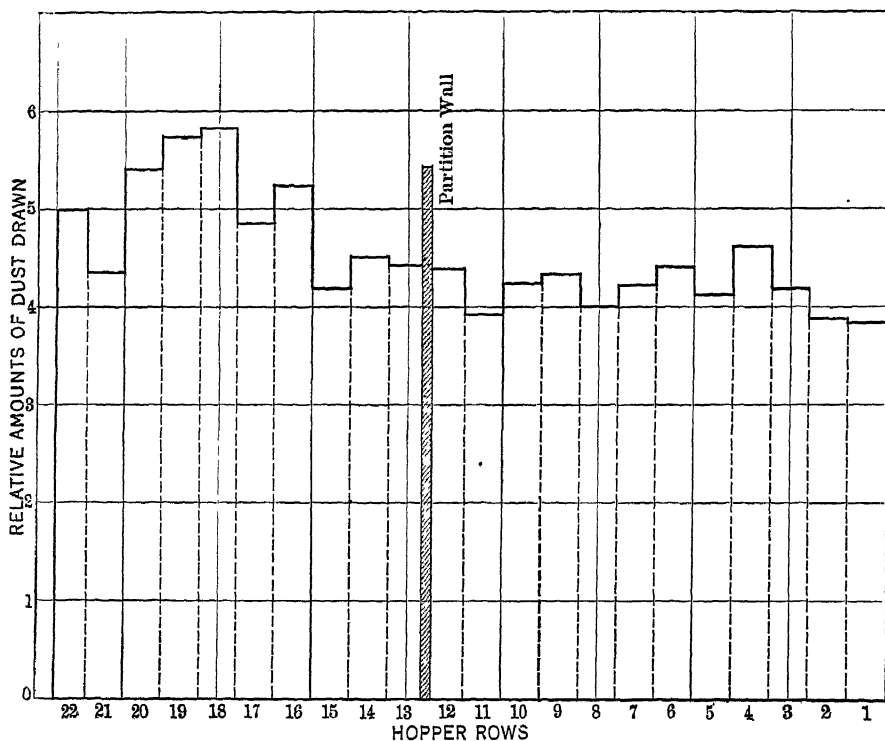
PLATE XXXV.—RELATIVE DEPOSITION OF DUST ACROSS DUST CHAMBER.

operation, from June 12, 1909, to Jan. 1, 1913. The increase in the amount of dust from hopper No. 1 is doubtless due to the flow of dust from the uptake when it is partially filled up from dumping cross-take dust into it.

Recent observations made by drawing out several wires from the central portion of both the upper and lower groups of wires show that in the first or lower group there is a light accumulation of dust on each wire which readily falls off when gently rapped. A cross-section through this accumulated dust is wedge shaped, the point of the wedge facing the current of gas being almost as thin as a knife edge. The depth of the

PLATE XXXVI.—PLOT SHOWING DUST DEPOSITION IN BLAST-FURNACE FLUE.

Per Cent	Tons
100.00	Total 86795.4
0.99	865.3
0.48	414.0
0.45	388.6
0.45	393.8
0.48	415.1
0.71	617.6
0.62	541.8
0.80	691.0
0.83	722.6
1.09	942.5
0.98	803.9
1.17	1018.3
1.23	1071.4
1.49	1290.0
1.98	1676.7
2.51	2180.6
2.86	2483.4
3.24	2810.1
2.69	2388.6
3.32	2884.1
3.26	2826.2
3.23	2801.5
3.78	3285.0
4.17	3616.2
4.05	3518.1
4.36	3783.7
4.41	3823.4
4.13	3585.0
3.94	3419.2
4.72	4097.0
5.10	4427.1
4.49	3900.0
4.13	3583.1
4.74	4117.0
4.97	4312.0
4.71	4088.3
3.54	3068.6

PER CENT OF TOTAL DUST DRAWN

HOPPER PLAN

Uptake

Total 4653.9 100.0

B.F. No. 1 — 1095.3 — 24.2
B.F. No. 2 — 932.4 — 20.6
B.F. No. 3 — 523.5 — 11.5
B.F. No. 4 — 942.6 — 20.8
B.F. No. 5 — 1040.1 — 22.9

Furnace Days
Per Cent of Total

wedge is from 1 to 2 in. and the back in which the wire is located is about $\frac{3}{8}$ in. thick. In the upper section of wires, the overall diameter of dust on wire is about 0.25 in., after shaking. These dimensions differ little, if any, from similar dimensions taken a year or more ago, which indicates that an indefinite increase in the permanent accumulation of dust on the wires is not to be feared.

X. TEMPERATURES AND DRAFTS.

The average temperature of the gases entering the wire-baffled dust chamber, exclusive of reverberatory gases, is about 325° F. = 163° C. The temperature at the upper end of dust chamber just after entering the connecting flue averages about 275° F. = 135° C. In the connecting flue, just before it branches to enter the chimney, the average temperature is 325° F. = 163° C. This includes reverberatory gases.

Temperatures are also taken at about the central point of the blast-furnace flue. These vary considerably depending upon the number of furnaces in operation and the conditions under which they are working. A fair average temperature when three furnaces are running is about 375° F. = 191° C.

These temperatures are taken with Bristol pyrometers having seven-day charts. Recording draft gauges are connected at the same points where temperature readings are taken. A typical set of draft and temperature curves covering the months of February and March, 1911, are presented in Plate XXXVII.

TABLE XI.—*Draft Readings on New Flue System.*

LOCATION OF READING	Elevation	Temperature in Flue F.°	Temperature of Atmosphere F.°	DRAFT READINGS IN WATER	
				Impact Tube	Static Tube
A—Blast furnace.........	3,338	391	80	1 12	1.16
B—MacDougall	3,361	419	80	0.94	0.98
D—Cross-take.........	3,413	358	80	0 88	0.97
D—Cross-take..........	3,413	346	70	0.94	1 01
L—Connecting flue.	3,413	312	70	2 20	2.26
U—Near chimney..	3,570	310	70	1.80	1 84

Special draft readings taken on the new flue system at points A, B, and D (see Plate XIII) and at two points on the connecting flue, one of which was near dust chamber, designated as L, and the other near chimney, designated as U, are presented in Table XI.

Simultaneous observations were made at points A, B, and D and at D, L, and U. During this time there were three blast furnaces, nine
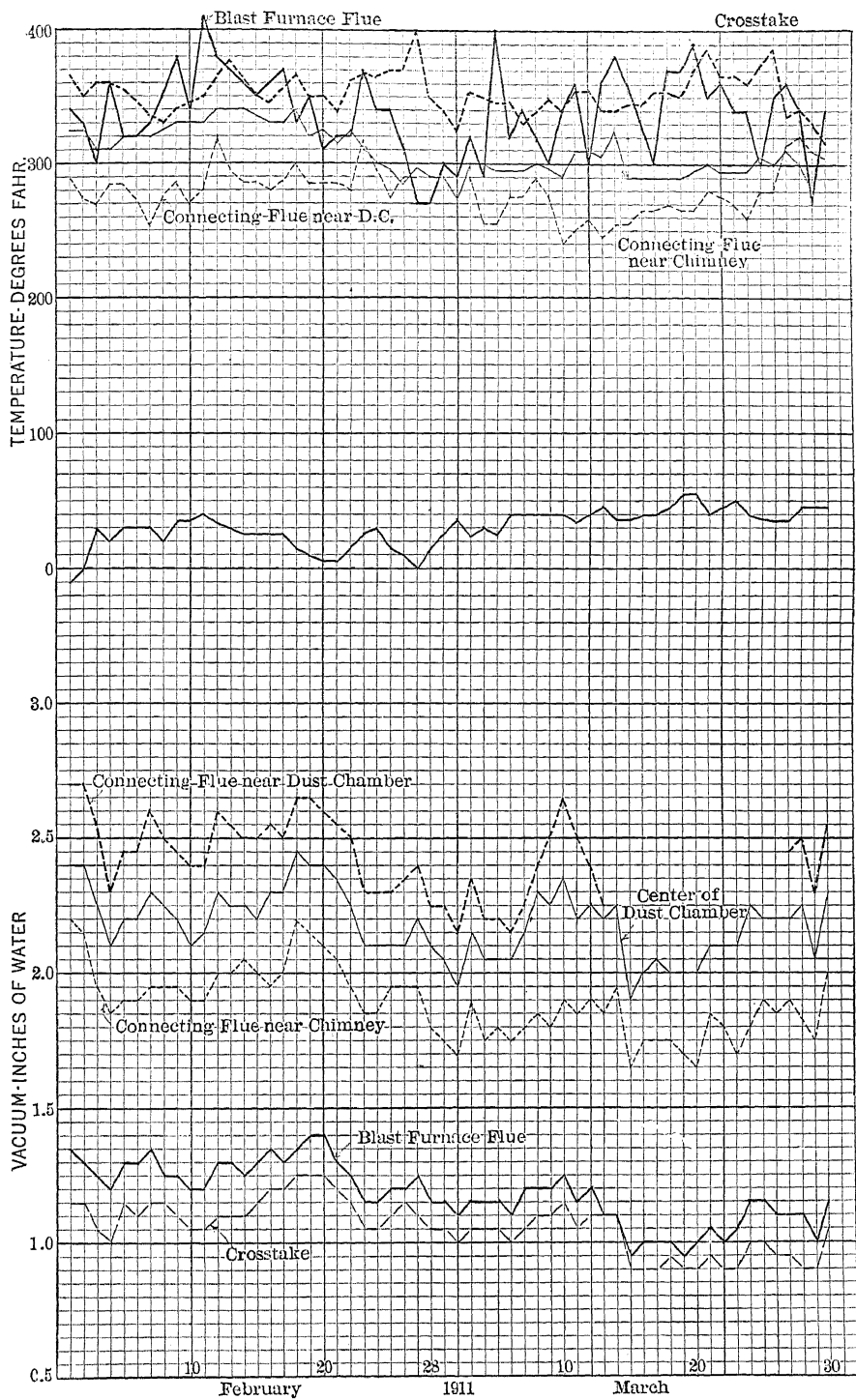
PLATE XXXVII.—DRAFT AND TEMPERATURE CURVES.   199

MacDougalls, three 12-ft. converters and three reverberatories in opera-
tion. Gases from two of the reverberatories were being bypassed around
the main dust chamber. In taking these readings, Pitot-tube elements
were used, the impact- and static-pressure ends being read independently.

Using these observations, the total draft energy on the MacDougall
furnaces is calculated, and also the loss in draft through the various
sections of the flue system.

The lower hearth of the MacDougall furnaces is at an elevation of 3,313,
while the top of chimney is at an elevation of 4,054 ft.

	Difference in Elevation Ft.	Average Temperature F.°	Draft Loss In. Water
Lower hearth of MacDougalls to cross-take	100	460	1 40
Cross-take through wire baffles to connecting flue.	0	329	1.26
Connecting flue near dust chamber to near chimney	157	311	0 25
Connecting flue near chimney to top of chimney..	484	300	0 06
Totals..	741	....	2 97

It should be explained that the loss through the wire-baffled dust
chamber shown in the above tabulation is higher than normal, because
of the fact that the upper section of wires had not been shaken for about
four months and also that there was an accumulation of several feet of
dust in the lower group of wires.

A comparison of draft readings at similar points on the old and new
systems may be of interest. Readings, of course, vary rather widely,
depending upon furnace and other conditions, but in general are about
as follows:

	DRAFT READINGS—IN. WATER			
	Blast Furnace Flue	MacDougall Flue	Reverb. Flue Near Furnaces	On Flue Near Chimney
Old system..	0.50	0.25	0 90	0.60
New system.	1.15	0 90	1 50	1.85

XI. VELOCITIES AND VOLUMES.

During March and April, 1911, tests were made to determine the
velocity and volume of gases passing through the various parts of the
new flue system. From these tests also, the amount of gas per furnace
and per ton of cupreous material treated has been calculated.

Pitot tubes were used in making velocity determinations, nine impact

points being inserted into the flue, representing as many equal rectangular areas. Bristol pyrometers were used for obtaining temperatures, and other instruments and methods were the same as have been described earlier in this paper.

Fig.1
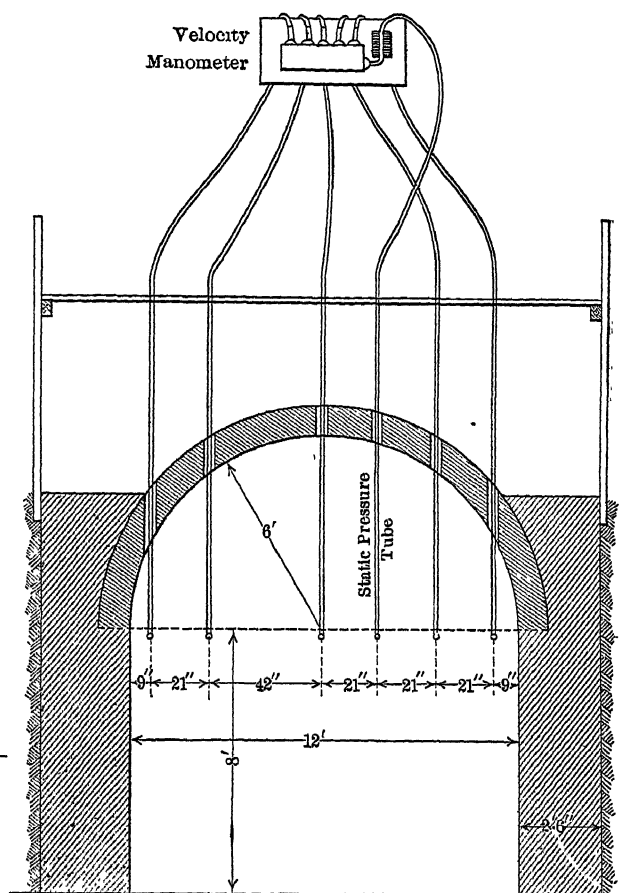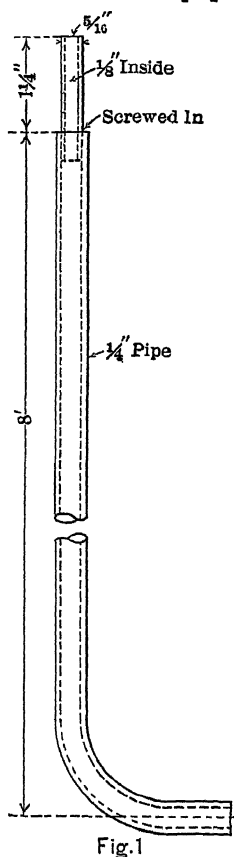
Fig.2

PLATE XXXVIII.—CONSTRUCTION AND ARRANGEMENT OF PITOT TUBES IN REVER-BERATORY FLUE.

During this test the reverberatory gases were bypassed around the dust chamber through the old No. 3 flue, in which velocity readings were taken with five Pitot points as indicated on Plate XXXVIII.

The furnaces referred to in the tables below are of the following sizes

Blast furnaces, 56 by 180 in. at tuyeres.
MacDougall furnaces, 14 ft. 6 in. inside of lining; six hearths.
Reverberatories, gas fired, 42 ft. 4 in. by 15 ft. 9 in.
Converters, 12 ft. upright.

TABLE XII.—*Velocities, Volumes, Average Temperatures and Weights of Gases.*

Location See Plate XIII	DATE, 1911	No. and Kind of Furnaces	Average Temperature F°	Clear Area of Flue Sq Ft	Vel in Ft. per Sec.	Vol at Observed Temperature Cu. Ft per Min.	Pounds Gas per Min.	Pounds Gas per Furnace per Min
A	Apr. 6–8...	4 B. F.	345	401 6	17.26	415,960	18,510	4,630
B	Apr. 6–8	10 MacD.	352	169.0	17 50	253,500	10,980	1,100
C	Apr. 6–8 .	5 Conv.	303	78 5	50 37	213,630	9,840	1,970
G	Mch. 21–24	2 Rev.	496	152 5	43 34	396,700	14,530	7,270
D_1	Apr. 6–8...	4 B. F. 10 MacD. 5 Conv.	331	636	21 80	859,300	38,190	. . . .
E_1	Mch. 21–24 .	4 B. F. 6 MacD. 5 Conv.	286	977	15 51	901,400	42,280	. . . .
F	Mch. 21–24 .	4 B. F. 6 MacD. 5 Conv. 2 Rev.	311	977	21 03	1,234,900	56,110	. . . .
E_2	Apr. 10 . .	4 B. F. 10 MacD. 5 Conv.	290	977	15.54	910,950	42,270	. . .
D_2	Apr. 10	do.	322	636	22.74	867,760	38,700	.

Tests A, B, and C on the individual department flues show a total of 39,330 lb. of gas per minute. The gases from these flues unite in the cross-take on which a simultaneous test was made showing 38,190 lb. per min., thus checking within about 3 per cent.

Tests D_1 and E_1 taken at the inlet and outlet end respectively of the main dust chamber cannot be compared directly, since they were made on different days, and a different number of furnaces were in operation. Tests D_2 and E_2 may be compared since they were taken on the same day and under the same conditions.

Lb.

D_2—Gas entering main dust chamber through cross-take flue. 38,700

E_2—Gas leaving main dust chamber through connecting flue. 42,270

Leakage of air into main dust chamber 3,570

Leakage in per cent. of gas entering main dust chamber. . . . . 9.2 per cent.

The temperature of the atmosphere on the day of the test was 54° F. Calculating the temperature which would result from mixing 3,570 lb. of air at 54° F. with 38,700 lb. of gas (considered as air) at 322° F., we obtain a temperature of 299°. Since the air leaving the dust chamber was at a temperature 290°, the loss due to radiation was 9° F. Summarizing we have:

Temperature of gases entering chamber through cross-take flue	322° F
Temperature of gases leaving chamber through connecting flue	290° F.
Temperature of outside air..... ...	54° F
Drop in temperature of gases in passing through chamber	32° F
Drop in temperature of gases caused by leakage of cold air..	23° F.
Drop in temperature of gases caused by radiation	9° F.
Per cent. of total drop due to leakage of cold air	71 9
Per cent. of total drop due to radiation....	28 1

A further check on the accuracy of the tests may be made by comparing the sum of tests E_1 and G with test F. Test E_1, which includes blast furnace, MacDougall and converter gases plus test G, which covers reverberatory gases only, show a total of 56,810 lb. per minute. Test F on the connecting flue near the chimney, where all furnace gases were combined, shows a total of 56,110 lb. gas per minute, being a difference of about 1.25 per cent.

Based on the amounts of gas for the various department flues, as presented in Table XII, Table XIII is presented to show the volume and weight of gas for one furnace of each kind when operating on the new flue system.

TABLE XIII.—*New Flue System.*

Amount of Gas per Furnace and per Ton of Charge.

KIND OF FURNACE	Observed Temperature Corresponding to Given Volume F.°	RATE PER MIN.		RATE PER 24 HR.		Average Tons Charged	PER TON OF CHARGE	
		Cu. Ft.	Lb.	Cu. Ft.	Lb.		Cu. Ft.	Lb.
Blast.. .. .	345	103,990	4,630	149,745,600	6,667,200	a391 6	384,900	17,000
MacDougall..	352	25,350	1,100	36,504,000	1,584,000	70 7	516,300	22,400
Converter...	303	42,730	1,970	61,531,200	2,836,800	b32.0	1,922,900	88,600
Reverberatory..	496	198,350	7,270	285,624,000	10,468,800	188 5	1,515,200	55,500

a Includes flux but does not include fuel.
b Tons copper produced per converter day.

Similar values for the old flue system are shown in Table XIV.

TABLE XIV.—*Old Flue System. Amount of Gas per Furnace and Per Ton of Charge.*

KIND OF FURNACE	Observed Temperature Corresponding to Given Volume F.°	RATE PER MIN.		RATE PER 24 HR.		Average Tons Charged	PER TON OF CHARGE	
		Cu. Ft.	Lb.	Cu. Ft.	Lb.		Cu. Ft.	Lb.
Blast .. .	600 .	89,400	3,030	128,736,000	4,363,200	*a*417 0	308,720	10,46(
MacDougall..	562	9,140	320	13,161,600	460,800	38 2	344,540	12,060
Converter ..	338	24,900	1,120	35,856,000	1,612,800	*b*30 9	1,160,390	52,190
Reverberatory.	1,000	78,680	1,930	113,299,200	2,779,200	207 3	546,550	13,410

a Includes flux but does not include fuel.
b Tons copper produced per converter-day.

A parallel comparison of the old and new systems is shown in Table XV.

TABLE XV.   *Comparison of Old and New Systems.*

KIND OF FURNACE	OBSERVED TEMP. OF GAS F.°		POUNDS PER MINUTE			POUNDS PER TON OF CHARGE		
	Old	New	Old	New	Excess of New Per Cent.	Old	New	Excess of New Per Cent.
Blast	600	345	3,030	4,630	52 8	10,460	17,000	62 5
MacDougall	562	352	320	1,100	243 0	12,060	22,400	85 7
Converter. ..	338	303	1,120	1,970	75 9	52,190	88,600	69 8
Reverberatory .	1,000	496	1,930	7,270	276 0	13,410	55,500	314.0

The points of determination of the gas volumes for the two systems were as follows:

Blast Furnaces:
   Old system—On old flues above skyline, four furnaces in operation.
   New system—On blast-furnace flue between No. 1 furnace and uptake, four furnaces in operation.

MacDougall Furnaces:
   Old system—Directly over No. 2 furnace, test on one furnace only.
   New system—On MacDougall flue near furnaces, gas measured from seven furnaces.

Converters:
   Old and new systems measured at same point on steel flue over roof.

Reverberatory Furnaces:
   Old system—On uptake east of old stone dust chamber, two furnaces in operation.
   New system—On old No. 3 flue south of skyline tracks, two furnaces in operation.

The large excess of gas from the MacDougall furnaces under present conditions is due not only to a better draft, but to a very much improved arrangement of flues, which greatly reduces the draft hindrances, as already pointed out.

The cause of the large increase in the weight of the waste gas from the reverberatory furnaces is not so easily explained, but, in addition to better draft conditions, it is believed that the quantities given include a big leakage of cold air which came in through the old stone chamber. This belief is strengthened by the results of some temperature readings, which were taken on the reverberatory waste gases at a time when the gases from three furnaces were passing through the old stone dust chamber. These readings gave an average temperature tip of 1,057° F. just before entering the stone chamber and a temperature of 605° F. after leaving the chamber.

The measurement on volume of reverberatory gas on the old system was made on the uptake from the underground flue before it entered the stone dust chamber, and for that reason includes no leakage of cold air except what comes in at the furnace, and consequently is less in volume and of a higher temperature.

Based on the quantities of gas given in the tables above, the approximate velocities through the various parts of the new system as it is being operated at present are as follows:

	Ft. per Sec
Blast-furnace flue......................	17 3
MacDougall furnace flue.	17.5
Cross-take flue	21 8
Main dust chamber.......	4 0
Connecting flue near chimney...	21 0
In chimney....	10 0

The velocity of 4 ft. per second in the main dust chamber is based on the full area, clear of dust and wires. The actual velocity through the wires varies with the amount of dust in the flue but will probably average from 20 per cent. to 25 per cent. higher.

XII. Tests on the New Flue System.

In September, 1911, a series of tests was started to determine the volume and solid contents of gases passing through the new flue system. These tests were carried out under the supervision of G. N. Libby, to whom we are indebted for the following description of apparatus and methods employed in taking the samples:

"The apparatus used in the determination of the suspended and condensable material in the flue gases is shown in Plate XXXIX. Samples were taken in each case simultaneously at opposite points in the flue 8 ft. from the east and west sides, by means of a 10 ft. by 0.5 in. glass tube inserted horizontally through openings in the flue wall and supported by angle irons fastened in these openings. The inner end of the glass tube was bent at right angles and a platinum tip 5 mm. internal diameter fused in the same. This 5-mm. opening was placed in the flue facing the

flow of gases.   Connected to the outer end of the tube in series were
(1) two 1-liter wash bottles each containing about 250 c.c. of distilled
water,   (2) a glass tube 10 in. by 0.5 in. filled with glass beads and   (3)
finally, a 'dust collector' with a paper filter.   This latter was kept warm
by an electric light bulb placed near it, in order to keep condensing
water from clogging it up.   Only the slight heat necessary to keep the
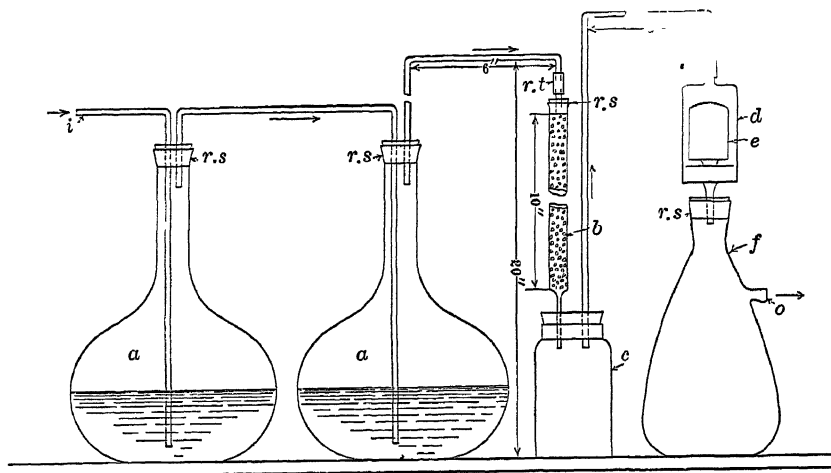
a, a. 250=cc. Florence flasks, about half full of distilled water.
b. Filter tube with glass beads.
c. 8-oz. wide-mouthed bottle.
d. Glass shell for filter paper cartridge.   (Shell was inclosed in pasteboard box with
one 60-watt lamp to avoid condensation.)
e. Extraction cartridge (packed in cotton).
f. Filter flask.
i. Inlet to system.
o. Outlet of system.
r.s. Rubber stoppers.
r.t. Rubber tubing.
Arrows indicate direction of flow of gases through filter system.
All connections made air tight.
Glass tubing is ⅛ in. diameter.

PLATE XXXIX.—GAS SAMPLING APPARATUS.

gases above the dew point was used.   It was regulated by insulating the
lamp with varying amounts of glass wool.
   "Aspiration was effected through this system by the displacement of
water in two large ammonia drums of about 10 cu. ft. capacity.   One
was being filled while the other was emptying and aspirating, thus giving
a continuous suction.   Glass gauge tubes were fitted in the side of each
drum and the drum calibrated by placing it on a tested pair of scales,
filling it with water to the zero mark on the gauge and allowing the water
to run from the drum 0.10 cu. ft. at a time, by weight, and marking the
changed height of water on the gauge glass.

"Endeavor was made to keep the rate of aspiration such that the gases would be drawn through the 5-mm. opening at the same speed as the gases passing. This, of course, could only be approximated, but was close enough for all practical purposes.

"The volume of the sample was determined by reading the number of cubic feet of water displaced and correcting to standard conditions. A thermometer placed in the drum gave the temperature, and a manometer just after the filter gave the pressure. These were read after the aspiration of each drum.

"A test of the gas after passing through the apparatus was made, and it was found that only a trace of As_2O_3 remained.

"After the sampling was finished the apparatus was disconnected from the aspirators, the flue tube washed off on the outside, and the whole taken to the laboratory where the flue tube, the wash bottles, the bead tube, and beads were washed thoroughly and the matter obtained, together with the water-soluble content of the filter, placed in a large evaporating dish and evaporated slowly to dryness on the steam bath at about 90° C. This matter was then all carefully scraped up, placed in a small weighing bottle, dried at 95–97° C. for 24 hr. and weighed. This weight, plus the difference in weight of the filter before and after aspiration dried at 95–97° C., was called total collected matter. On account of the H_2SO_4 content of the sample it was, of course, impossible to drive off all moisture at the temperature used in drying, and this probably accounts for most of the difference which exists between the total collected matter weighed and that found by chemical analysis when calculated with its maximum possible oxygen content."

Samples were taken at two places on the connecting flue. The first set was obtained at the top of the first 30 per cent. grade north of the main dust chamber. The reverberatory gases enter the connecting flue through a bypass further up and were therefore not included in these samples.

The second set of samples taken from Oct. 5 to 10, 1911, was obtained from each of the two branches just before entering the chimney. The results of this latter set only will be presented, for the reason that all smelter gases were combined before passing the point where the samples were taken.

Observations for velocity and volume of gases were made simultaneously with taking the samples. In addition to the regular velocity readings taken during the time covered by this series of tests, several determinations were made on the average velocity of the gases over the whole flue area, by taking observations at a large number of points in the cross-section. A chart showing the results of one of these determinations is presented in Plate XL. This chart also shows the depth of dust in the

flue, which was determined by sounding from the roof, and the location of the Pitot points used for regular velocity determinations. It will be noted that the velocity in the central portion of the flue is lower than at the sides. This may be due to the proximity of the dividing point where

Hole No.	1	2	3	4	5	6	7	8	9	10	11	12
Av. Vel. Head	0.100	0.110	0.110	0.100	0.082	0.096	0.072	0.074	0.090	0.110	0.114	0.104
1	0.09	0.10	0.11	0.11	0.08	0.10	0.07	0.06	0.10	0.11	0.11	0.11
2	0.10	0.11	0.11	0.10	0.09	0.10	0.07	0.08	0.09	0.11	0.11	0.11
3	0.11	0.12	0.11	0.09	0.09	0.09	0.07	0.09	0.09	0.11	0.12	0.11
4	0.10	0.11	0.11	0.10	0.08	0.10	0.08	0.08	0.09	0.11	0.12	0.11
5	0.10	0.11	0.11	0.10	0.07	0.09	0.07	0.06	0.08	0.11	0.11	0.08

Flue Dust
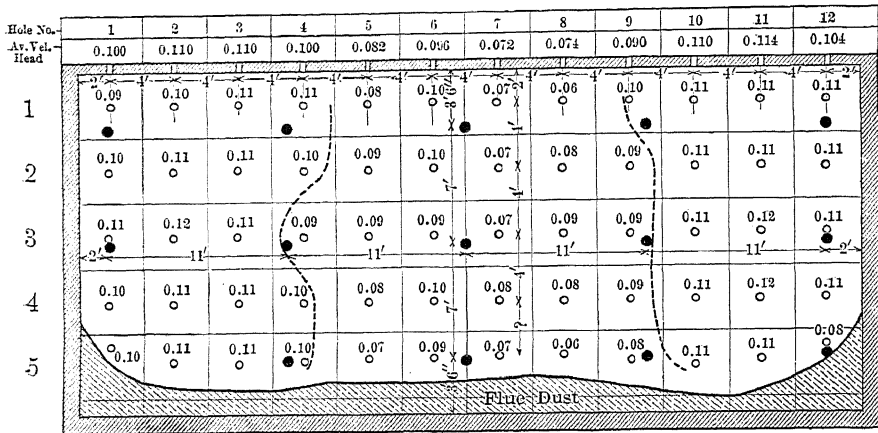
1. Av. V. H. for West 16 ft. = 0.103.
   Av. V. H. for Middle 16 ft. = 0.078.
   Av. V. H. for East 16 ft. = 0.107.
   Av. V. H. for Entire Section = 0.096.
2. Av. V. H. for West 16 ft. = 0.105.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.105.
   Av. V. H. for Entire Section = 0.098.
3. Av. V. H. for West 16 ft. = 0.108.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.108.
   Av. V. H. for Entire Section = 0.100.
4. Av. V. H. for West 16 ft. = 0.105.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.108.
   Av. V. H. for Entire Section = 0.099.
5. Av. V. H. for West 16 ft. = 0.105.
   Av. V. H. for Middle 16 ft. = 0.073.
   Av. V. H. for East 16 ft. = 0.095.
   Av. V. H. for Entire Section = 0.091.
Av. V. H. for Entire Flue = 0.097.
Av. V. H. for West 16 ft. of Flue = 0.105.
Av. V. H. for Middle 16 ft. of Flue = 0.081.
Av. V. H. for East 16 ft. of Flue = 0.105.
Broken Line Connects Approximate Points of Average Velocity.
All Velocity Head Readings Given in Inches H₂O.
● Indicates Points Where Regular Readings Were Taken.

PLATE XL.—CROSS-SECTION OF FLUE AT STATION 42.5 FT. SOUTH OF "Y," SHOWING VELOCITY HEAD READINGS.

the flue branches into the two chimney openings. It was found that the average velocity as determined by this special test checked closely with the results of the regular velocity observations.

As explained by Mr. Libby, the samples were aspirated from the flue through glass tubes inserted through the side walls, and extending approximately to the central point of the flue. The total amount of the dust sample with this type of apparatus is necessarily small, but the results of the various tests are very uniform.

Samples were taken at two points in each branch of the flue, the upper point being 9 ft. below the roof and the lower point 8 ft. 6 in. above the floor, the two branches of the flue being designated as East and West respectively. Three samples were taken at each point, making 12 in all.

During these tests the following furnaces were in operation: two blast furnaces; two reverberatory furnaces; eight MacDougall furnaces; and 2.6 converters (average). The average results of the three samples taken at any given point, together with the general average for the whole series are shown in Table XVI.

TABLE XVI.

SAMPLE POINT	MAIN FLUE GASES		Average Cu. Ft. of Sample Stand. Cond.	Average Solids Collected Grams	CALCULATED SOLID CONTENTS OF ALL FLUE GASES PER 24 HOURS			
	Velocity Ft. per Second	Cu. Ft. per 24 Hours Stand. Cond.			Solid Matter Lb.	Cu Lb.	aAg Oz.	aAu Oz.
Upper E........ ...	18 63	901,816,800	180 4	4 566	51,220	311	.....	.....
Lower E......	19 34	960,591,840	184.4	4.225	48,630	347	...	.....
Upper W.	18.63	901,816,800	170.8	4 672	54,900	358	.....	... .
Lower W.	19 34	960,591,840	183.6	4.174	48,010	373	.....	..... .
AVERAGE. ...	18 98	931,204,320	179.8	4.409	50,690	347	28.51	0 202

a These values obtained by proportion, the sample not being of sufficient weight to make direct determinations.

The average amount of new cupreous material charged to furnaces per 24 hr. during this period contained 181,220 lb. of copper.

The loss of copper in escaping gases in per cent. of original copper charged to furnaces was therefore 0.19 per cent.

An average partial analysis of the suspended and condensable material contained in gases sampled is presented in Table XVII.

Tests on the old flue gases by the use of wire baffles in the experimental brick flue, indicated that 89.7 per cent. of all copper entering the flue with blast-furnace gases, and 96.2 per cent. of all copper entering with MacDougall gases, was recovered in the flue.

TABLE XVII.— *Partial Analysis of Suspended and Condensable Matter in Gases.*

Constituent.	Per Cent. of Total.
Free H_2SO_4	22 23
Silica	2 30
Copper	0 70
Fe_2O_3, Al_2O_3	7 08
Sulphur	6 67
Sb_2O_3	1 47
Bi_2O_3	0 81
PbO	0 49
CaO	0.18
ZnO	3 31
O to combine with sulphur (calculated)	10.23

Based on recoveries from the new wire-baffled dust chamber, and estimates on the amount of copper in the dust contained in the connecting flue, and the amount escaping from the chimney, we find that 94.2 per cent. of all copper entering the chamber is recovered from it, showing a remarkably close confirmation of test results.

Calculations based on the amounts of material in the flue system, as determined by weights drawn, and by estimates on dust and fume in the connecting flue and escaping gases, are presented in Table XVIII.

TABLE XVIII.—*Percentage Distribution of Material in Flue System.*

	Weight	Copper	SiO_2
Blast-furnace flue	39 3	41 4	47 3
MacDougall furnace flue	8 5	11 3	11 0
Cross-take flue	7 8	12.9	8 8
Main dust chamber	28 9	32 5	30 7
Connecting flue	1 8	0 7	0.7
Stack discharge	13 7	1.2	1 5
TOTALS	100.0	100.0	100.0

Production of Arsenic.

Up to the present time no effort has been made to recover arsenic from flue dust, as the market for this material, until recently, has not justified the necessary outlay for its production, but we have just completed a plant which will handle about 20 tons of the dust per day.

For the present only the dust recovered from the upper end of the main dust chamber will be handled in this plant. Later it may be found desirable to increase the recovery of arsenic by the admission of cold air to the chamber, provision for which was made in the original design, as already explained.

XIII. CONCLUSION.

Comparing the losses from the old flue system, as determined by tests with the experimental brick flue, with the losses on the new system, as shown by the tests which have just been described, and allowing for the relative amounts of copper produced at that time and at present, the following differences are noted:

	Loss per 24 Hr.		
	Copper Lb	Silver Oz	Gold Oz.
Old system	3,775	105 94	0 712
New system....	347	28 51	0 202
Difference saved	3,428	77 43	0 510

Assuming a recovery of 95 per cent. of these values, and taking a net value of 10c. per pound for the copper, 50c. per ounce for the silver and $20 per ounce for the gold, it is seen that the new flue system has effected a saving of $372.18 per day, or $130,263 per year.

In addition to this saving it is expected that a substantial revenue will be derived from the treatment of the more arsenical dust, in the recently completed arsenic plant.

In considering the large expenditure for the installation of the elaborate flue system just described, it should also be borne in mind that a reduction in losses of flue dust was not the only object in view. The low elevation at which the gases were discharged from the old chimney was a source of annoyance from smoke complaints. The overtaxed condition of the old flues and chimney prevented the operation of certain furnaces at full capacity, and for the same reason the smoke and fume around the furnaces materially interfered with the efficiency of labor. Furthermore, dust chambers were inadequate and their poor arrangement made the handling of flue dust expensive and troublesome.

All these defects have been eliminated, and the anticipated efficiency of the new flue system has been fully realized.

DISCUSSION.

PROF. JOSEPH W. RICHARDS, South Bethlehem, Pa.:—Mr. Chairman, I would like to call attention to one valuable feature of the paper which may interest some: I have worked a good deal with the Pitot tube, and have always had difficulty with what is called the static end of that apparatus, so much so that I became almost hopeless of securing good results with it. I find here, however, what is stated to be a perfectly satisfactory static end for the Pitot tube, satisfactory under probably all conditions. I congratulate Mr. Klepinger on the design of that tube and the consequent improvement he has made in the technical measurement of the volumes of gases. I would also like to ask if the total volume coming out of the chimney per day could not be computed by chemical analyses of the SO and SO$_2$ in the gases, and calculation on the basis of the known weight of sulphur going out in a day; so that the Pitot tube measurements could be checked by a method independent of measurements in the flues.

J. H. KLEPINGER:—I think the volume of gases could be determined in that way. Those of a chemical turn of mind would go about it so, but I am not enough of a chemist to follow that method, and consequently I always look at the mechanical way to obtain the results.

HOWARD N. EAVENSON, Gary, W. Va. (communication to the Secretary*):—The writer has been much interested in this very able paper presented by Messrs. Goodale and Klepinger, and particularly so in those portions of it relating to the measurement of the velocity through the flue and the settling of the dust in the flue chamber. The form of static pressure end developed for the Pitot tube will undoubtedly give very good results with the velocities measured, but with very high velocities—say above 10,000 ft. per minute—it is questionable whether the instrument will be substantial enough to resist the pressure unless it be made so heavy as to materially obstruct the flow of gases. The averaging manometer is an excellent device and the results obtained by it will probably be within the limits of accuracy of measurements up to velocities of about 6,000 ft. per minute.

In order to complete the data in the paper, the writer would suggest that if any traverses of the experimental flue were made, showing the different velocities in various parts of its area, that these be included in the final publication of the paper.

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* Received Sept. 24, 1913.

The curves shown in Plate IX of the relative efficiency of the various schemes used for dust arresting are evidently based on the average velocity in the experimental flue of 440 ft. per minute given in Table IV. Were any tests made of any of these schemes, using higher or lower velocities than this? The wire baffles are evidently the most efficient and economical of all the methods tested, at this velocity, and probably so at any other velocity, but undoubtedly tests 32 and 34, with the open flue, would have been much more efficient at much lower velocities, as would probably test 37. The enormous volume of gases to be handled would of course have precluded the use of any scheme involving a very low velocity—say 100 to 200 ft. per minute; but probably the various tests made by the authors have enabled them to form an opinion as to the effect of lower velocities on the efficiency of the various methods tested.

From the data given on p. 597, the average weight of dust per cubic foot in the gases is slightly over 48 lb. Would this be a fair average weight of dust in all experiments?

Great credit is due both of the authors for the interesting presentation of the data obtained in their various tests and in the successful working out of their design of flue system and chimney, both in its details and in its entirety.

J. H. KLEPINGER, Great Falls, Mont. (communication to the Secretary*):—Mr. Eavenson refers to Plate IX, the curves in which show the relative efficiency of the various dust-arresting schemes tested, and raises the question as to the possible change in efficiency if tests had been made, in which the velocities in the experimental flue were higher or lower than those for which the results are presented.

As suggested by Mr. Eavenson, the large volume of gases to be handled made it impracticable to obtain low velocities, on account of the extremely large cross-sectional area of flue required. This point had been considered previous to making the final tests, at which time it was decided that the velocity of gases should not exceed 500 ft. per minute. The velocity in the experimental flue was therefore made to conform to this value as nearly as the conditions and the apparatus at hand would permit. In this connection we wish to correct Mr. Eavenson's assumption that the curves in Plate IX represent the results of tests at the uniform velocity of 440 ft. per minute. On the contrary, they show the actual weight of dust collected per unit volume of gas handled under the actual conditions existing in the experimental flue, which conditions are given in Table III, and show

that the velocity varied from 568 ft. per minute for test No. 32, to 266 ft. per minute for test No. 35.

It should be explained further, however, and in answer to a question raised by Mr. Eavenson, that the velocities in the experimental flue, as given in Table III, are all calculated on the assumption of a full sectional area of the flue, free of any baffles or other obstructions, the calculations being based on volume determinations in the 22-in. inlet pipe, no velocity readings or traverses having been made in the experimental flue.

The assumption of a full sectional area of the flue is of course incorrect, the actual free area being somewhat less, and varying with the different types of baffles used, with the amount of dust deposited on the floor, which in many cases amounted to several inches, and with the accumulation of dust on the baffles. In the absence of any definite data on the amount of these various obstructions, we thought it best for purposes of comparison to base velocities on the full sectional area, as stated above. We did make some rough estimates as to the probable velocity through the free area in the flue for the various tests, which indicated a more uniform condition than that shown in the table, the maximum difference being not more than 150 ft. per minute; but, as these were only estimates, we did not feel justified in presenting them.

On account of the difference in velocity in the experimental flue for the various tests, it might be said that we are not justified in making the direct comparisons of efficiency that are shown on Plate IX; and we believe it to be true, as pointed out by Mr. Eavenson, that if tests 32 and 34 had been made at considerably lower velocities they would have shown a higher efficiency. This belief is verified by tests which we made when using the steel settling chamber, referred to briefly in Article 2 of Section IV of our paper. With this apparatus we ran tests having velocities as low as 30 ft. per minute. Recoveries at this velocity did not differ materially from those at 100 ft. per minute, which may indicate that practically all dust was settled at the latter velocity; but when the velocity was increased to 200 ft. per minute there was a decided falling off in settlement of dust. We do not, however, believe that the tests made with the small steel settling chamber are fairly representative of what would be obtained in a large flue. Even if it were possible to make a flue of proper section to bring down the velocity, it would be necessary to have it of comparatively great length to effect a reasonably complete settlement of the dust. In all our tests, the results pointed to the necessity of some obstruction or surface being present, against which the dust could

impinge or come in contact, in order to effect a settlement. This condition seems to be more important than the reduction in velocity, and for this reason we believe that a flue of large sectional area, without any baffles or obstructions, would be even less efficient than the small experimental flue in which our tests were made.

Assuming for the moment that the efficiencies shown by the curves in Plate IX are not comparable on account of the differences in velocities, we can nevertheless make a very direct comparison of the open flue against baffles, in tests 35 and 38, where certain sections of the flue were equipped with baffles and others left free and open. Since the same gas passed through each of the sections for a given test, the velocity in the open sections must have been even lower than that in the sections containing baffles, where the free area was certainly less; yet there is a very marked difference in the settlement of dust.

This is pointed out, not that any one doubts the efficiency of baffles, but simply to emphasize the rate of settlement by the two methods, and the relative lengths of flue that would be required to effect equal recoveries.

With reference to the average weight of dust in the gases, per cubic foot, it may be said that this is practically constant for gases from a given type of furnace. Test No. 39 was made on No. 3 flue, which carried MacDougall-furnace gases; while tests Nos. 40 and 41 were made on the No. 1 flue, which at that time was handling blast-furnace gases only. For this reason it would not be fair to obtain the average by combining all the values given in the table referred to on page 597. This table was presented simply to show the difference in weight per cubic foot of the dust in the first and last sections of the flue. The true average weight of the dust for a given test would be greater than the arithmetical average of the values given, for the reason that by far the greatest volume of dust is collected in the first few sections of the flue, as may be seen by referring to Tables IV, VI, and VII.

Determination of Gases in Smelter Flues; and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont.

BY EDGAR M. DUNN, ANACONDA, MONT.

(Butte Meeting, August, 1913)

OUTLINE.

PART I. DETERMINATION OF GASES IN SMELTER FLUES.

General considerations.
Regular methods for carbon monoxide, oxygen, nitrogen, moisture.
Sulphur oxides—Methods of Lunge and Hempel tried, and either
    modified or discarded, with reasons therefor.
Gravimetric method developed at Anaconda.
Description of apparatus used.
Brief notes on Richter and Hawley methods for the direct deter-
    mination of sulphur trioxide.
Carbon dioxide, with complete notes on same.
Arsenic trioxide.
Table of percentages of flue gas constituents found at a Western
    smelter.
Some final notations as to the value of flue gas analysis, in con-
    nection with an intelligent log of furnace operations.

PART II. NOTES ON THE DETERMINATION OF DUST LOSSES AT THE
        WASHOE REDUCTION WORKS, ANACONDA, MONT.

Discussion of problem, with description of flue system.
Various places and methods for determinations of velocity.
Instruments used.
Procedure in making determinations for velocity.
Soundings of cross-section for depths of dust.
Calculation of velocities and areas.
Daily volume of gas, and plant operations during tests.
Place for dust determinations.
Apparatus and methods of sampling.
Results obtained, with table of daily amount of dust losses.
Analysis of dusts for values, from various sampling stations.
Complete analysis of composite sample of stack emanation.
Last analysis totaled to show form in which present.
Discussion of recovery as a commercial proposition.
References on Pitot tube and calculations.

PART I.—DETERMINATION OF GASES IN SMELTER FLUES.

IN 1907, upon arriving in Anaconda to take up work in the testing department of the Washoe Reduction Works, the following problem was met at the car step, so to speak: complete and exact analysis of gases, not only at the stack, but also in all departmental flues. Taken up to employ time not otherwise needed for regular testing department work, instructions were extremely liberal for a commercial plant, to the effect that accuracy should not be sacrificed to speed. In December, 1907, the work was temporarily transferred to the Great Falls Smelter, and some of the methods here given were there worked out. Differences in smelting operations at Great Falls and Anaconda, however, necessitated further investigations and the development of a new method for SO_3 determinations at the latter plant, where work was resumed in July, 1908, and completed about a year later.

At the time of inception of the work, data (not only on percentages, but even on methods) were found to be meager in the extreme, for one or the other of two reasons: either entire lack of experiment at a large proportion of smelters, or disinclination on the part of the managements to divulge data of any description pertaining to either results or methods.

In making determinations of gaseous constituents of flues, two methods are open to consideration: 1, the drawing out of a sample, and collection of the same as a whole, to be removed to the laboratory for later analysis; and 2, the absorption *in situ* on the flue of one certain constituent, disregarding others for the time being, or perhaps combining two or three in one determination. The first is generally the more rapid, but it is also the more inaccurate, method of analysis, even with special type absorption apparatus, being the more largely subject to error, both of reading, and also because of leakage during complicated manipulation in the laboratory. The amount used for analysis must also, because of limit of container, be small for the first method, and an average gas is very seldom obtained, necessitating for a fair average result a large number of determinations. Consequently, when we balance the time spent in a dozen determinations of short-time samples as against the time employed in making two or three long-time determinations of various constituents *in situ*, we are not so certain in our statement as to the relative speed of the two methods. Moreover, the method of analysis in the laboratory of 100 cc. of flue gas is satisfactory only when constituents thus determined are present in considerable quantity, results being simply approximate when determining such compounds as sulphur dioxide, sulphur trioxide, carbon monoxide and dioxide, and arsenic trioxide and water vapor, none of which are carried in *all* smelter flues in amounts in excess of (or even approaching) 1 per cent. by volume. Special condi-

tions at certain smelters may raise the percentage of water vapor to that amount; for instance, employment of wet fine concentrates for roaster feed, or use of briquettes while still wet in the blast, with poorly dried coarse concentrates. Generally speaking, however, five or six of the above-named determinations should be made independently of each other, or at least in special apparatus, *in situ* on the flue; but for carbon monoxide (never present with good practice in any smelter flue in considerable quantity), the difficulties in connection with the removal of other constituents in large quantities are so great as to indicate the rather unsatisfactory (from accuracy standpoint, because of trace quantity) method of its determination in the laboratory by absorption pipette of either ammoniacal or acid cuprous chloride, after removal of interfering gases in suitable reagents.

Determinations of oxygen, carbon monoxide, and nitrogen (residual gas after all other constituents are removed) were made by standard methods, and require no description here. All samples were collected over mercury and were analyzed by means of the Hempel mercury burette. Twelve 16-oz. bottles, filled with mercury and supported in a strong, specially designed, two-deck rack, were used for taking samples, and analyses of the last 10 (first one largely and second one slightly contaminated by air in aspirating tube) were used as a basis for results.

Nor was the method used for water vapor new in any respect. The gases were drawn through a hot filter of glass wool in the flue end of the aspirating tube, and into weighed bulbs containing sulphuric acid. Results obtained in this way are somewhat too high, but are more than approximately correct.

Sulphur Dioxide and Trioxide.

Two standard methods were available for SO_2 and SO_3 at the time of starting our work. Briefly, Lunge's method consists in drawing gases through standardized alkali solution, with phenolphthalein indicator, determining both SO_2 and SO_3 as total acidity, as indicated by formation of the respective sulphide and sulphate reactions:

$$4 \ NaOH + SO_2 + SO_3 = Na_2SO_3 + Na_2SO_4 + 2 \ H_2O;$$
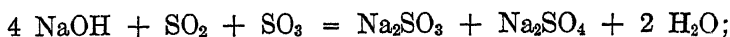
then a determination by the "Reich" method with standardized iodine solution of the amount of SO_2 present in the gas by the following reaction:

$$SO_2 + I_2 + 2 \ H_2O = H_2SO_4 + 2 \ HI;$$
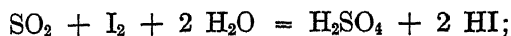
the SO_2 factor from the iodine determination is then subtracted from the total as determined by the alkali, with a remainder of SO_3. We found Lunge's method (volumetric entirely) accurate only when the flue gases contained no substances of a nature contaminating to either

iodine or alkali value, which is a rare condition, and we have been very much interested by recently published conclusions of a similar nature. At practically all smelter flues some one or more of the adverse conditions are met. These conditions are the following: 1, presence of arsenic, selenium, or tellurium in the ores, which may vitiate any flue gas as far as the Reich test is concerned, if present in sufficient quantity, rendering results approximate rather than exact, by too high a figure for SO_2 from attack of iodine by oxides of these elements. 2, the presence of H_2S in roaster flue gas, where a wet sulphide is slowly brought to roasting temperature. This last, if operative, will also cause too high an SO_2 figure, from the consequent attack of the iodine by the H_2S. We do not believe, however, that this is often the cause, though positive affirmation of same was at one time made to us by a prominent engineer. At least, we have never found, in frequently testing various conditions, hydrogen sulphide present in roaster gases (certainly never more than trace), and simply offer it here as a possible (not probable) explanation of the too high SO_2 figures so frequently found to be the actual result. 3, the fact that reverberatory flue gases (coal fired) contain variable amounts of hydrocarbons due to imperfection of combustion at certain stages of their operation; a fact hitherto unrecognized as to the action on iodine, causing by the mutual reaction too high a figure for SO_2 content of gas. We have several times obtained appreciable quantities of the red copper acetylene compound by passing reverberatory gases through ammoniacal cuprous chloride. The presence of acetylene points, of course, to other hydrocarbons of a similar character. , This hydrocarbon factor seems to be absent in flues from gas-fired reverberatories, probably due to more perfect combustion control. As to oil-fired reverberatories, we have no data.

All of the above-mentioned factors tend to raise the SO_2 content of the gas in the Reich part of the Lunge method, and consequently, even if the total acidity (as determined by alkali) remains correct, yet the figures finally obtained for SO_3 (by subtracting too high SO_2) are correspondingly too low. Suppose, however, that the alkali titration is also open to errors that cause total acidity ($SO_2 + SO_3$) to be low. Of course, the method as outlined by Lunge becomes then not even approximate, the SO_3 figure being negative. To illustrate, under perfect conditions the following little arithmetical equation is true:

$$(SO_2 + SO_3) \text{ minus } SO_2 \text{ equals } SO_3.$$

In practice, however, the following is more often the case:
$(SO_2 + SO_3—\text{too low}) \text{ minus } (SO_2—\text{too high}) \text{ equals } \textit{less than nothing.}$

In an article that appeared the early part of this year (1913), in connection with a direct method for the determination of SO_3 by absorption, E. Richter (abstracted in the *Journal of the Society of Chemical Industry,*

Mar. 31, 1913) recognizes the failings of the Lunge-Reich method, the abstract (we have not read the original article) commencing as follows: "The determination of sulphur dioxide and trioxide in furnace gases by Lunge's and Reich's method does not give satisfactory results; either the reactions with iodine and potassium hydroxide do not take place in the theoretical proportions, or else substances are present which combine with a higher proportion of iodine than [does] sulphur dioxide; at any rate the results for trioxide often work out as a minus quantity." Richter's alternative is a method for determination of SO_3 direct, by absorption in a condensing tube packed in a freezing mixture, a method capable, we believe, of giving good results. A similar abstract of Mr. Richter's paper appeared in the *Engineering and Mining Journal* of May 10, 1913.

The last-named magazine (vol. xciv, pp. 987 and 988, Nov. 23, 1912) also gives an excellent article on sulphur dioxide and trioxide determination, by F. G. Hawley, chief chemist at Cananea, Mexico. Mr. Hawley emphasizes the cause of error in the alkali titration for total $SO_2 + SO_3$; though our own experience differs somewhat from his. We quote from him as follows: "An objection to this method is that it cannot be used on furnace gases because of the action of CO_2 on the phenolphthalein. Even in roaster gases, for which it is recommended, there is likely to be some CO_2 formed by the oxidation of small amounts of organic matter usually present in ores, as well as from small amounts of carbonates which may evolve CO_2 during roasting. Sulphates of the heavy metals are always present in the fume and will react acid to phenolphthalein, and thus affect the result."

Now we never tried the Lunge-Reich method anywhere else than at Anaconda, and Mr. Hawley may have ores of a character somewhat different from ours. The alkali titration works fairly well on our roaster and converter gases, the figures obtained by titration checking closely with gravimetric amounts obtained by later oxidation of sulphites to sulphates and precipitation of all sulphates with $BaCl_2$. Of course, in all of our work we eliminate all dust and practically all fume by slow aspiration and a hot (flue temperature) glass-wool filter in the flue end of our aspirating tube. But we feed to our roasters large amounts of carbonate in the shape of fine lime rock (later utilized in slag formation in the reverberatories and fed to the MacDougalls to insure better mixing), and yet the CO_2 evolution of the gas is but slightly greater than atmospheric amount, the critical temperature of calcium carbonate not being attained in roasting furnaces. Should ores be employed containing carbonates with lower critical temperature than that of calcium carbonate, the method would be worthless for roaster as well as for other flues using carbonaceous fuel. When CO_2 is present, there are three anhydrides

attacking the alkali instead of the two (SO_2 and SO_3), the CO_2 also re-acting. The salt formed by the last, however, is alkaline, and remains inert as to acid action on phenolphthalein until the last stages of neutralization, when the original alkaline hydroxide is all gone with formation of carbonate, sulphite, and sulphate, the first named, however, with an alkaline reaction toward the indicator. Further passage of CO_2, SO_2, and SO_3 causes an attack of the carbonate by the two latter, but before they have finally changed the carbonate to sulphite and sulphate (which would be a true titration of the original alkali to the two salts formed by SO_2 and SO_3), the accumulation of CO_2 in the solution (both from flue absorption and from liberation from carbonate in the solution itself) swings the phenolphthalein reaction to acid, indicating an end point for SO_2 and SO_3 on NaOH before it has been reached, and causing *low* results for total SO_2 and SO_3 by this volumetric method. We early discarded the Lunge-Reich process.

The second method at our disposal in 1908 was the variation given by Hempel and mentioned by Mr. Hawley in his article. Briefly, it is a combination of volumetric and gravimetric methods, the gases (CO_2, SO_2, and SO_3) being passed into standardized iodine as in the Reich test, and the SO_2 content being determined by the iodine factor, and this SO_2 figure being subtracted in terms of SO_3 from the total SO_3 figure (caused by both SO_2 and SO_3) later determined by gravimetric precipitation of acidified solution with $BaCl_2$. This method is open to all the objections previously mentioned in connection with the Reich test, the practical result being: SO_2 plus SO_3 (gravimetric and accurate) minus SO_2 (volumetric and too high) equals too low SO_3 (occasionally *nil*). CO_2, however, does not interfere or affect results in any way. We used this method in specially designed apparatus of our own (which will be later described), but with varying results, and finally evolved a method entirely gravimetric in character.

This last method, adopted by us after thorough experimenting, employs simultaneous determinations with two sets of apparatus, the determination for total SO_2 and SO_3 being made by passing gases through iodine solution, to convert SO_2 to SO_3 (at the same time collecting all original SO_3 unchanged). A gravimetric determination with $BaCl_2$ gives total $SO_3 + SO_2$, the latter in terms of SO_3. The original SO_3 has been simultaneously determined on the same gas by passing the gases through the second set of apparatus, the solution for absorption in this second case being weak $BaCl_2$ and from 10 to 15 per cent. of HCl. Mr. Hawley mentions this method in his article, and we suspect our original notes (which were furnished in 1909 and 1910 to a large number of Western smelters) caused him to look further, because of incompleteness of data therein, and finally evolve his scheme of filtering out the SO_3 by means

of a moistened filter, with later titration of contents of filter, which is a variation of the dust method abstracted in the *Journal of the Society of Chemical Industry*, vol. xxii, p. 1016.

We find the following opinion on the direct determination of SO_3 by means of $BaCl_2$ in the presence of SO_2 in Lunge's *Sulphuric Acid and Alkali*, vol. i, p. 317: "It was found that sulphur trioxide cannot, as Scheurer Kestner supposed, be absorbed and estimated by $BaCl_2$, because even chemically pure SO_2 with $BaCl_2$ in the presence of oxygen or atmospheric air, at once gives a precipitate of $BaSO_4$." We checked Lunge's work, but found the greater part of his precipitate to be, not $BaSO_4$, but $BaSO_3$, and soluble in HCl. As $BaSO_3$ is theoretically impossible in the presence of HCl, we made several experiments with SO_2 generated by means of dilute HCl and $NaHSO_3$, passing same with air through acidulated $BaCl_2$. We found repeatedly: 1, no $BaSO_3$ formed in any of a series of four flasks; 2, faint cloudiness in first flask, which was trace of $BaSO_4$; 3, relatively large to small amounts of SO_2 dissolved in all four flasks, which were later boiled out entirely without precipitation of any $BaSO_4$ from the last three flasks, and without changing visible amount of precipitate in first flask; 4, the amount of $BaSO_4$ in first flask after boiling thoroughly to expel SO_2 and to collect $BaSO_4$ for filtering was (maximum) 0.0065 g. $BaSO_4$, or somewhat less, for eight runs, using in all cases over 100 times as much SO_2 as we aspirate in a regular flue determination. The facts that all four flasks contained SO_2 dissolved therein, but none of the last three were precipitated by immediate boiling out of same; while the first flask was not precipitated by boiling, but only by passage of original SO_2 + air; force the conclusion that the trace of $BaSO_4$ found in the first flask was caused, not by SO_2, but by SO_3, formed between the generator and the first flask by admixture with air, *before* entrance to first flask.

Mr. Hawley objects to the boiling out of the SO_2, because of oxidation to SO_3, and consequent high results for the latter, and he ascribes oxidation found by him to atmospheric oxygen. With substitution of passage of CO_2 for removal of dissolved SO_2 (rather than a quick boiling out of same), he characterizes the method as accurate, though tedious, because of generation and passage of the CO_2. We found the method (without the tedious CO_2 passage and with a simple boiling) consistently correct; and at another of the Western smelters where *no* SO_3 was present in their stack gases, the results were persistently *nil*, though the stack gases did carry a much larger amount of SO_2 than is present at Anaconda. However, if Mr. Hawley boiled out the contained SO_2 over a free flame, or in direct sunlight, his conclusions as to high results are in agreement with our own results; for, while in position on the flue or while boiling off the SO_2 in the laboratory, the apparatus must be carefully protected

from any other than dull, diffused light, as we later found that the action of direct sunlight, or the light of a free flame at any point in the process between start of aspiration and final complete breaking up and boiling off of contained H_2SO_3, was destruction to the retarding action of HCl in the solution. In spite of the HCl, sunlight or free flame light acts as a catalyzer, causing quick and considerable oxidation of H_2SO_3 to H_2SO_4, with consequent high results for SO_3 content of flue gas by this method. Of course, boiling off of the SO_2 should be accomplished immediately after aspiration of gas, as standing between aspiration and release of contained SO_2 also causes results to run somewhat high, even in diffused light.

Apparatus.—The apparatus employed by us was as follows:

Set I of the apparatus, Fig. 1, consists of a wooden box 26 in. long by 9 in. wide. One end is left open for a distance of 9 in., and this compartment is covered with a board 9 by 9.5 in., which contains four slots 1 in. wide for holding the necks of the 4-oz. flasks firmly in place. The rest of the box is sided up with boards 4.75 in. high, and is to contain the first aspirator bottle, and such pieces of apparatus as may be carried conveniently therein. The end pieces of the box are so cut down near the top (12 in. high) as to make handles for carrying the apparatus.

Four 4-oz. flasks are used in making the analysis. These should be supplied with *tightly fitting* two-hole rubber corks.

Two large glass bottles (about 3 liters capacity) are used as aspirating apparatus, for the double purpose of drawing the gas from the flue and also measuring the amount of gas sample taken. The first bottle should be fitted with a three-hole rubber cork; the second requires a two-hole cork. Rubber tubing 10 or 12 ft. long may be used for connecting the two aspirating bottles. The first of the 4-oz. flasks, *B*, is connected with the flue by a one-piece bent tube, *A*, the long arm of which is about 3 ft. long, and the short arm about 9 in. long, which allows the long arm to be inserted in the flue about 2 ft. The flue end contains a loosely packed filter of glass wool, for stoppage of dust.

The four flasks are connected in series; *C*, *D*, and *E* being connected by glass tubes bent in two right-angle bends, the longer arm going nearly to the bottom of each flask (the inlet tube), and its end being nearly closed for slow aspirating purposes.

The shorter arm (outlet tube) is of such a length that, when the flask into which it goes contains 150 cc. of solution, the end of the tube (blown out bulb shaped, and containing one or more fine holes) will be at such a distance above the solution that, when running gas, the beads contained in this short arm will be kept well moistened with solution. The short arm should *not* dip in the solution, but its height above surface

26"

9"

1"

9"

TOP VIEW

12"

4¾"

SECTIONAL ELEVATION.

ANALYSIS BOX.

Out ← ← In

B

Connections of first flask.

Three tubes connecting
four flasks.

K J L T G

F

H

Water replaced
by gas.

D

Open to atmosphere

C

A

B

E

L

M

Air replaced
by water.

Flue Roof

APPARATUS IN PLACE ON FLUE.
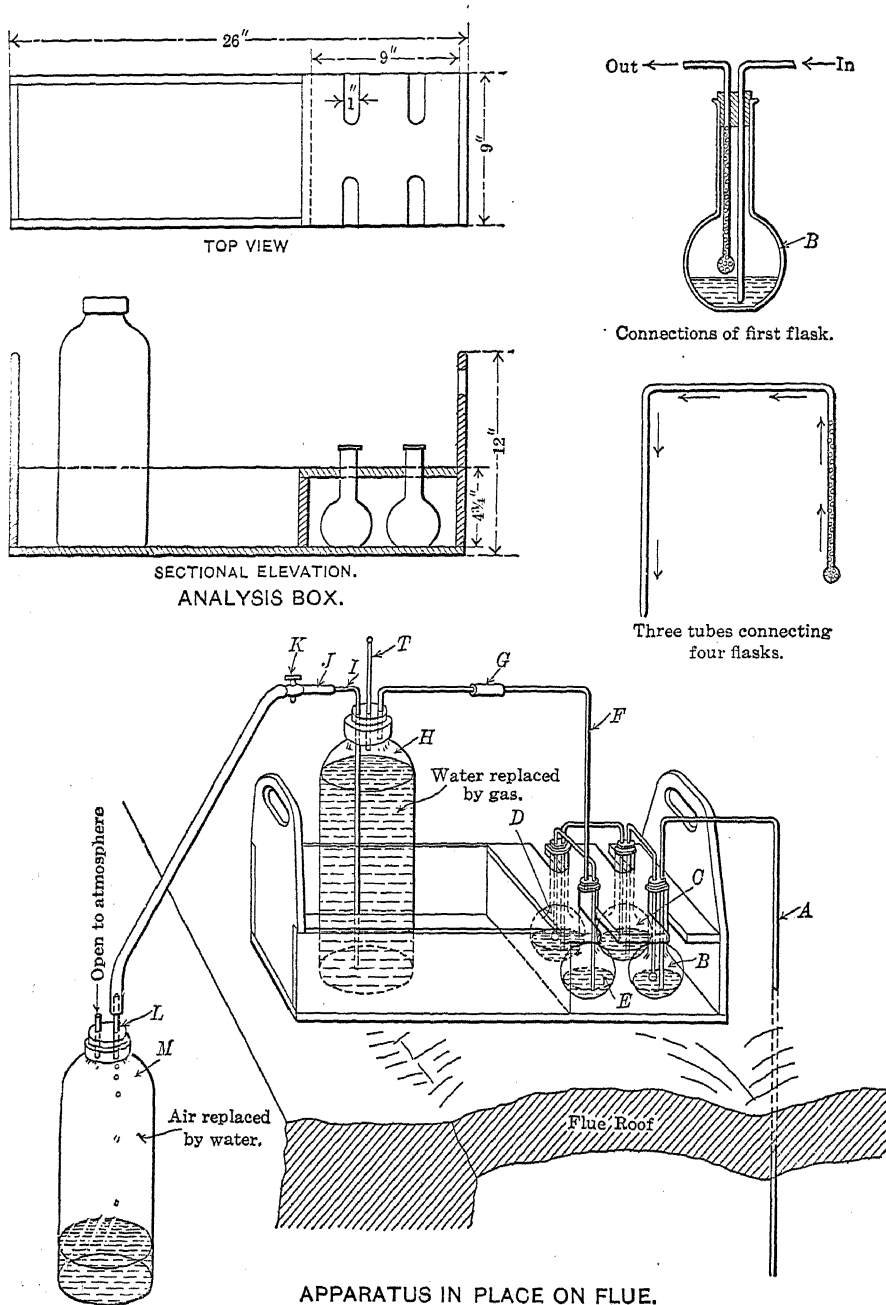
FIG. 1.—APPARATUS FOR DETERMINATION OF SULPHUR OXIDES.

of same will be governed by the speed of aspirating—the faster the aspiration, the higher the short arm, and *vice versa*.

The second hole of the fourth flask, *E*, is connected with one of the three holes in the cork of the first aspirator bottle by means of a two-arm glass tube, *F*, which is connected at about the middle by a heavy piece of rubber gas tubing, *G*, which is well wired to the glass to insure against possible leakage of gas. Both ends of glass tube *F* pass barely through the respective corks of the fourth 4-oz. flask and of the first aspirator bottle. The rubber connection is not necessary (*F* being one piece instead of two pieces), but should a one-piece tube be used, chances of breakage from over-stiffness of apparatus are much increased.

In the second hole of the cork of the first aspirator bottle, *H*, a thermometer, *T* (preferably Fahrenheit for closer reading), should be inserted. By this means the temperature of the gases aspirated may be ascertained.

Through the third hole of the cork in *H* a bent glass tube, *I*, passes to the bottom of the bottle, and is bent at its upper end at a right angle (or more) to connect with a rubber tube, *J*, 10 or 12 ft. long, through which the water runs from *H* to the second aspirator bottle, *M*, the top of which, when in action, should be some little distance below the bottom of the first aspirator bottle. This rubber tube is provided with a stop cock, *K*, for regulating the speed, and connects at its lower end with a piece of glass tubing, *L*, which penetrates one hole of the two-hole cork in the second aspirator bottle *M*, the other hole of which is left open. If preferred, *M* need not be corked, but left open.

Several precautions in taking the sample are of primal importance. All connections from *A* to *J* *must be* absolutely gas tight. For this reason all rubber connections between flasks are eliminated, also the one commonly used about the middle of tube *A*. Such elimination, of course, increases rigidity of apparatus, and consequently greater care must be employed in handling the same, to guard against breakage.

All holes in rubber corks (except bottle *M*) should be tightly fitted to glass tubing, and to insure loss against leakage all such fittings should be reinforced, after setting apparatus in place, by a liberal and careful application of rubber cement.

Air must be carefully excluded from the aspirating tube *A*. To this end, the tube should be inserted at least 2 ft. in the flue, and should be carefully packed at entrance into flue with slag wool or waste.

Before passing from methods for the oxides of sulphur to carbon dioxide, the processes of Messrs. Richter and Hawley for SO_3 determination must be considered. Either the absorption after cooling (Richter), or the filtration through a moistened filter (Hawley), should give good results. Both employ the passage of atmospheric oxygen for removal

of SO_2 from the sponge which has just absorbed total SO_3 and more or less SO_2, but in one (Hawley) the presence of contaminating dust can have no effect, because of titration rather than gravimetric determination. The same variation might easily be applied to Richter's method. With regard to these sulphur gases, two former ideas of a mistaken character are now exploded: 1, that SO_3 (with its ordinary tremendous affinity for water) is easy of absorption, for we now know that it is the hardest gas of any encountered in smelter analysis to stop in water solution, probably because of the minuteness of its quantity, the SO_3 in each bubble of gas being surrounded by larger overcoats of more or less insoluble gases; and 2, that the oxidation of H_2SO_3 to H_2SO_4 is a rapid process, for we have learned that the reverse is true, and that even this slow process can be still further hindered by exclusion of catalyzers and introduction of retarding agents. The accuracy of both Richter's and Hawley's methods is dependent on this latter fact. The H_2SO_3 is driven out by passage of air, which requires considerable time, and yet no oxidation to H_2SO_4 occurs.

Carbon Dioxide.

For the determination of CO_2 in flue gases we followed Pettenkofer's method for determination of CO_2 in air, adapting the process to our own apparatus, and changing the strength of solutions to meet requirements of gases in the various flues.

With two sets of apparatus, the three constituents SO_2, CO_2, and SO_3 can be determined simultaneously, using the first set for total SO_2+SO_3 (first two bottles) and for CO_2 (third and fourth bottles) and the second set for direct SO_3.

The method for CO_2 as adapted by us to flue conditions is here given more fully perhaps than is necessary, but with a view to illustration of our idea of tabulation of process notes.

Apparatus.—The apparatus used for the determination of CO_2 is the same as that employed for SO_2 and SO_3 (Set I). In addition, a small 4-oz. flask is fitted with a two-hole cork, through one hole of which passes a glass tube just through the cork. This tube is for connecting the flask with the flue tube, A, of Set I; the other hole of the cork is fitted with a tube which extends from the bottom of the flask up through the cork. Through this second tube the air when operating passes into the flask to the bottom and bubbles up through the flask full of strong NaOH solution. For purposes of reference call this flask X.

Solutions Required for Analysis.—The solution of NaOH for use in flask X need not be of exact strength, as it is simply for the purpose of removing CO_2 from air passed through the apparatus. Dissolve about 100 g. of NaOH in about 500 cc. of water.

Iodine Solution for Removing SO_2 and SO_3 from the Gases Aspirated.—
Dissolve about 10 g. of iodine and 15 g. of potassium iodide in from 100
to 200 cc. of distilled water. Shake until iodine is entirely dissolved and
dilute to 1 liter. If desired, N/10 iodine may be used.

Barium Hydroxide Solution for Absorbing the CO_2 in the Gases Aspirated.—
Dissolve about 14.8 g. of $Ba(OH)_2$ in 2 liters of distilled water, and
filter into the stock solution bottle. Mix thoroughly and cork bottle
tightly with a two-hole rubber stopper. Through the first hole of this
cork passes a long double bent glass tube to the bottom of the bottle
inside; the other end extending several inches below the bottom on the
outside. This siphon tube is clamped with a stopcock, and when not
running out the solution, the second hole of the rubber cork is also closed
with a tightly fitting glass rod. This $Ba(OH)_2$ solution should be stand-
ardized against the oxalic acid solution so that its value is known in pre-
cipitating power against both the standard oxalic acid solution and car-
bon dioxide. When running $Ba(OH)_2$ out of stock bottle, inflowing air
should be passed through flask X.

Standard Oxalic Acid Solution.—Place 100 cc. of boiling distilled water
in a No. 2 beaker, and add oxalic acid crystals until solution is saturated
hot. Cool the solution and dry the resulting oxalic acid crystals by re-
peatedly pressing between folds of filter paper, until the crystals are thor-
oughly dry, and pour like sand from one sheet of paper to another. Weigh
out exactly 5.6293 g. of this recrystallized oxalic acid, and dilute to 1
liter with distilled water, which is preferable freshly redistilled. It is
best to double the amounts of $C_2H_2O_4$ and H_2O, making 2 liters of
solution. After solution is attained place in a dried glass-stoppered bottle.
This solution is of such a strength that each cubic centimeter is equal
to 1 cc. of carbon dioxide, at standard conditions, in its neutralizing ac-
tion on barium hydroxide; or, in other words, 1 cc. $C_2H_2O_4 = 1$ cc. CO_2.
Its strength should be checked against standard Na_2CO_3 solution.

Phenolphthalein Indicator.—Weigh out 1 g. of phenolphthalein and place
in a glass-stoppered bottle of about 250 cc. capacity. Add 210 cc. of
strong ethyl alcohol and shake until dissolved.

Standardization of $Ba(OH)_2$ Solution.—Measure out 100 cc. of the
$Ba(OH)_2$ solution from a burette, add 5 drops of the phenolphthalein,
and titrate with the standard oxalic acid to decolorize the solution. The
$Ba(OH)_2$ will be found to be about half the strength of the oxalic acid,
or the 100 cc. of $Ba(OH)_2$ will require about 50 cc. of $C_2H_2O_4$. The $Ba(OH)_2$
crystals used are never of definite strength (having been more or less acted
on by the CO_2 in the air). In the determinations made the 100 cc. of
$Ba(OH)_2$ was found to be equal (phenolphthalein decolorized) to 48.75
cc. of $C_2H_2O_4$. Therefore, 1 cc. of $Ba(OH)_2$ equaled 0.4875 cc. of $C_2H_2O_4$
(of CO_2).

Method of Taking Sample.—Fill flask X with NaOH solution, and connect with the flue tube A (Set I) by means of a small piece of rubber tubing. Tube A is connected with the four absorption flasks (Set I), and the last of these flasks is connected by means of a long rubber tube with the air pump in the laboratory.

Draw air through the apparatus for 0.5 hr., by which means the air in the absorption flasks is replaced by air containing no carbon dioxide (which is removed by passage through NaOH solution in flask X). Disconnect and fill the first of the four absorption flasks with iodine solution to the 150-cc. mark. In the second absorption flask place about 130 cc. of redistilled water. Into the third and fourth flasks run in from a graduated burette, respectively, 150 cc. and 100 cc. of the $Ba(OH)_2$ solution. Do not dilute the $Ba(OH)_2$ in the fourth flask with ordinary distilled water (which contains trace of CO_2); either leave the volume of solution therein at 100 cc. or else dilute to the 150-cc. mark with redistilled water *free from* CO_2. Cork the absorption flasks tightly with the corks and bead tubes used in flue aspirating, and remove to the flue. Set up the apparatus (Set I) on the flue, just as in the determination of the sulphur gases, and aspirate for about 90 to 100 min. at the rate of about 2,000 cc. of water drawn from the first into the second aspirator bottle in the total elapsed time of aspiration. A faster rate of aspiration must be carefully avoided to prevent the passing through the first two absorption flasks of any SO_2 or SO_3, which would, of course, raise the percentage of CO_2.

The position of the outlet tubes from the first three absorption flasks must also be carefully watched. The outlet tube from the first flask (containing iodine) should be so set that the beads contained in the tube are at all times impregnated with the iodine to insure stoppage of the sulphur gases in the first flask. To check this fact, a *trace* of the iodine from the first flask should be allowed to run over into the second flask sufficient to impart a yellow tinge to the water contained therein. If at any time the aspiration should become .rapid enough to decolorize this yellow *tinge*, it shows that SO_2 is passing through the iodine into the $Ba(OH)_2$, and the determination is worthless. The outlet tube from the second flask should be slightly higher in position, so that the beads are simply moistened with the yellow-tinged water, and yet so that *none* of the solution finds its way into the third flask. Should this occur, the determination is also worthless, for two reasons: the percentage of CO_2 will be slightly raised by the attacking of the $Ba(OH)_2$ in the third flask by the solution that runs over from the second flask, and the end reaction used in later titration will be vitiated. The outlet tube from the third flask should be so set that the beads are constantly wet with the $Ba(OH)_2$ solution, a small amount of which should be allowed to pass into the fourth

flask. This precaution insures absorption of CO_2 in the third flask, largely, but never entirely, the fourth flask being here necessary. With large amounts of CO_2 a fifth flask must sometimes be used.

During the aspiration the formation of insoluble barium carbonate ($BaCO_3$) in the third flask and outlet tube tends to clog the beads and the stopcock which controls the aspiration will require constant watching to obtain a steady flow of gas.

At the end of the aspiration period close the stopcock and measure the water in the second aspirator bottle. This will require about 5 min., during which time the absorption apparatus and first aspirator should be left on the flue. Read the temperature of the gas in the first aspirator bottle and remove box to the laboratory without disconnecting flasks and first aspirator. Connect flue tube, A, with NaOH bottle, X, and aspirate slowly by means of the air pump for 10 min. to insure absorption of the sulphur gases standing in the outlet tube and over the iodine in the first absorption flask. Now raise the outlet tubes slightly in the first two absorption flasks and aspirate at about twice the former speed for 0.5 hr. to draw through the third flask any CO_2 which may have been held in physical solution in the first two absorption flasks. Cease aspirating and raise the corks in the absorption flasks, taking out the corks and tubes, and closing the third and fourth flasks with tight rubber stoppers, and proceed at once with the analysis.

Analysis of Samples.—To one of the third and fourth absorption flasks add 5 drops of the phenolphthalein indicator and titrate with the standard solution of oxalic acid ($C_2H_2O_4$) until the red solution is fading in one of the flasks. Cork and set aside, and titrate the solution in the other flask to the same point. Now just decolorize the solutions in both flasks and wash out the bead tube connections with the titrated solution. If a pink color develops in the flasks, again decolorize with the oxalic acid. Take the reading of the oxalic acid burette, and proceed to calculate results. If desired, the indicator may be added before flue aspiration.

Notes.—The solution ordinarily used in gas absorption pipettes for the removal of CO_2 is NaOH. This cannot be used in the flue determinations, for the reason that for absolute accuracy the CO_2 *must be removed from the solution as fast as absorbed,* so as to admit of the later titration of the excess of the alkaline solution with the standard acid.

If NaOH is employed, the aspiration of CO_2 results in formation of sodium carbonate, which is soluble and remains in solutions to be attacked later by the standard acid used in the titrating; while with $Ba(OH)_2$ as an absorption solution the CO_2 precipitates as barium carbonate, which is insoluble in all reagents with which it comes in contact (H_2O, $Ba(OH)_2$, and dilute $C_2H_2O_4$) and the titration with dilute $C_2H_2O_4$ is in no wise affected by its presence. If NaOH is to be employed the later titration

must be omitted; the indicator must be added to the alkali before aspiration; and the red solution must be decolorized in position on the flue by the aspiration of CO_2. This is a more rapid, but by no means a satisfactory, determination, for two reasons: 1, the end reaction between any alkali and CO_2 is not as quick and clean cut as that between an alkali and oxalic acid (especially true when phenolphthalein is used for an indicator; but true in some degree of any indicator); 2, this method of determination *loses* the CO_2 which is physically absorbed in the flasks of iodine and water (which are placed between the alkali and the flue for the purpose of absorbing the sulphur gases). This latter error is alone sufficient to condemn the determination, as close checking is impossible. We found 20 per cent. of total CO_2 held in first two flasks. Experience has proved that another very wide error may be introduced into an otherwise carefully performed determination, by the clogging of the apparatus by the $BaCO_3$ formed in the outlet bead tube from the third absorption flask as well as that in the third flask itself. In the previous methods employed for the determination of the sulphur gases, the products of absorption were soluble, and consequently evinced no desire to interfere with aspiration. This clogging with the insoluble $BaCO_3$ causes an increased suction to be required for drawing the gas through in the later stages of aspiration, and at the end of the aspiration it may be found that the gas in the first aspirator bottle is therefore under diminished pressure, and the water in the second aspirator does not therefore represent the true volume of gas aspirated from the flue.

In one of the preliminary tests of the method outlined about 1,500 cc. of water was drawn through in slightly over an hour, but in the latter part of that time the aspiration entirely ceased despite the fact that the stopcock on the tube between the two aspirator bottles was finally opened wide. The stopcock was then closed, the apparatus removed to the laboratory for analysis, and the back pressure was noticed to be large (by the inrush of air into the first aspirator when the stopcock was removed). This, of course, caused a figure much too large for the volume of gas aspirated, as shown by the fact of a 12 per cent. drop in volume percentage of CO_2 when the final results were obtained. The remedy, where the aspiration does not run smoothly and stoppage occurs, unless very near the end of the aspiration, is to throw out the determination, wash out the apparatus and begin over again. If desired, the back pressure can be measured, the error calculated, and the determination finished.

To prevent stoppage of apparatus, the tubes connecting the absorption flasks should be carefully washed out (between determinations) two or three times with dilute HCl. This dissolves the precipitated $BaCO_3$, and the excess of HCl must then be carefully washed out with distilled

water, or results for CO_2 will be too high (the HCl acting on the $Ba(OH)_2$ during aspiration). Eight to a dozen washings are required to eliminate the HCl.

Arsenic.

The analysis for arsenic vapor in smelter flues offers no great difficulty. This constituent is present in such small amounts that rapid aspiration, to obtain large volume, and analyzable amounts, is the first requisite. Consequently large carboys are employed for aspirating bottles. Experiment proved the impossibility of escape of As_2O_3 through the absorbing solution (water alone is excellent) even under a speed of 25 liters per hour, for the reason that the vapor readily condenses to a solid on cooling to boiling point of water. Two of the four bottles in our apparatus box are all that are necessary, though all four may be employed. Nor are beads in the connecting tubes required, their omission causing no loss of As_2O_3 while at the same time the speed of aspiration and ease of washing out tubes are thus facilitated. That part of the aspirating tube between the flue and the absorption bottles was found to contain a large proportion of the As_2O_3 (deposited therein by cooling), and it must therefore be thoroughly washed out with warm dilute NaOH solution. Analysis of the arsenic can be run by any standard method, but we prefer the so-called "sulphate method," substantially as given by A. H. Low in *Technical Methods of Ore Analysis*, as we have found it the most accurate of several tried.

Composition of Gases.

By courtesy of the management of one of the copper smelters where work was accomplished, we submit the following table of analyses for various flues:

CONSTITUENTS	Roaster Flue Per Cent.	Blast Flue Per Cent.	Converter Flue Per Cent.	Reverbera-tory Flue Per Cent.	Base of Main Stack Per Cent.
Sulphur dioxide	2.545	1.274	2 845	0 423	1.164
Sulphur trioxide . . .	0.275	0.086	0 0515	0.0044	0 0395
Carbon dioxide.	0.1136	6 493	0.2084	5.242	2.748
Water vapor.	2 784	3.490	1.061	3 869	2.834
Arsenic trioxide.	0.0073	0.0091	0.00073	0.0156	0.00465
Oxygen	14.02	10.18	12.04	10.37	11.88
Nitrogen.	81.18	78.13	83.64	79.57	80 73
TOTALS	100 9	99.7	99.8	99.5	99 4

All of the above are volume percentages at standard conditions—760 mm. and 0° C.

The only flue that showed certain trace of carbon monoxide was the reverberatory.

Selenium and tellurium oxides were found in both reverberatory and main flues, but were not determined quantitively, appearing incidentally during As_2O_3 aspirations. Traces of antimony were found throughout in all flues.

About a year later, with changed conditions at this same smelter, analyses of the sulphur oxides were repeated. Changes were three in number: 1, (slight) in tonnage; 2, in character of feed (considerably more zinc being present); and 3, the flues, which were in a poor shape because of leaks during the first series of analyses, had been put in nearly perfect condition.

In all flues, both departmental and main, the SO_2 content was higher, due to differences in both tonnage and condition of flues. On the other hand, the SO_3 percentage was lower, due to greater neutralizing action of the zinc.

At base of stack the SO_2 volume percentage was 1.734 per cent., a raise of about 50 per cent. over figures previously given, indicating leakage; while the SO_3 gave a beneficent drop of over a half, becoming 0.0159 per cent., indicating possible entire neutralization of SO_3 by slightly greater infusion of zinc in feed.

In closing, it should be emphasized that probably none of us realize fully what can be done at the average smelter, in the matter of actual saving of dollars and cents, by means of intelligent and accurate gas analysis. By "intelligent" we mean a minute and exact log of furnace operations, and a careful checking of tonnages, with analyses of materials fed and produced. Without these any analysis is simply a figure to satisfy curiosity, and indicates nothing for future practice. A case in point will be illuminating. At one plant where we worked, the orders were to keep entirely away from all operations, confining ourselves entirely to analysis. The SO_2 in blast-furnace gases showed at different times tremendous variations, and the average of a large number of determinations was reported as the content of the blast-furnace flue, but the figure carried no illumination as to cause of variation, which had to be ascribed roughly to differences in tonnage and feed. At a second smelter, the same variation was observed, but a careful study of conditions showed that, at times when the SO_2 was high, there was less coke fed to the furnaces than the supposed minimum amount required for smooth running. When the SO_2 was at its low ebb in the flue the coke was fed either in normal or excess of normal amount, and the sulphur was volatilized, *and not utilized as fuel.* The furnaces ran smoothly under either condition. The indicated result, the regular running of the furnaces

with less coke, with correct utilization of the sulphur fuel sometimes thrown away, and a consequent saving in good hard cash by the management, should forever remove "intelligent" gas analysis from the schedule of luxuries at the second plant.

Aside from considerations of indicated changes in practice, a minute log of operations often serves to clear up results of a contradictory character. We remember working in a converter flue on two succeeding days, when with a larger number of converters blowing the first day the flue content of SO_2 was considerably lower than that of the second day, when fewer converters were blowing, with conditions of feed and products identical, which certainly looks absurdly incorrect. But we fortunately had included in our log the stage of the blow of each converter while sample was being drawn from the flue, and a study of this confirmed the result obtained, for on the second day (higher SO_2 content) a majority of the converters were in the stage "white metal to copper;" while on the first day the preliminary, or "slagging," stage of operations predominated. We had previously (sampling every 15 min. of the blow of about 2.5 hr. at a single converter mouth) found that the preliminary stage (oxygen used in iron silicate formation) gives in round numbers 1.0 per cent. of SO_2 while the succeeding blow of white metal to copper evolves as high as 6.0 per cent. of SO_2, the oxygen being utilized in sulphur oxidation. Without a perfect log, flue results would in this instance have been discarded with disgust, but with the log for guidance, results at first glance impossible were predicated.

Many other examples might be cited as to the importance of analyses of gases at the average smelting plant, but space does not permit. In addition to value for determining leakages between furnaces and stack emanations, or for checking up metallurgical balances of various parts of the plant, one other actual experience in "detection of crime" will be mentioned. Recently, at Anaconda, balances of the arsenic plant showed loss of some As_2O_3, back into the main flue with which both sets (first and second refining furnaces) are connected after the products are settled in two separate lines of kitchens. The product settled in both sets of kitchens differed by only a few per cent. in $As_2 O_3$ content. Which set of furnaces and kitchens was causative of loss and what is the remedy?

Analysis for As_2O_3 at exhausted end of both sets of kitchens (either one would do it when comparing its loss with the total) caught the culprit, and a careful log of the two operations showed (draft the same for both) too high a heat and too great an admission of air through the refining furnace and kitchens, indicating changes in practice necessary to prevent loss.

Three final words: 1, Accuracy of analysis; 2, Minute log of all operations; and 3, Diligent study of results for contraindications of practice

observed; first, chemistry; second, bookkeeping; third, metallurgy; these three form a combination too strong to be neglected.

PART II.—NOTES ON DETERMINATION OF DUST LOSSES AT THE WASHOE REDUCTION WORKS, ANACONDA, MONT.

In the winter of 1910–11 an exhaustive investigation was carried on by us, with the purpose of determining as closely as possible, incidentally the velocity of our smoke stream, with accurate figures for the daily amount of stack emanation, and ultimately the amounts and character of dusts carried in this emanation. Could the gross amount of smoke passing the stack be handled by any of the processes (more or less at that time in experimental stages) either in use or being installed at various smaller smelters? The gross amount being approximately, of course, known to us at that time, if such installations, with known enormous initial expense, were made, would the amount of dust and fume, if recovered successfully, pay anything toward this large installation cost; and, if not, would the values be sufficient to guarantee upkeep and daily operation of this smoke plant? Or would dust values be so small as to render both items of expense largely chargeable to smoke alone, tending to wipe out the margin of profit on smelting costs at the prevailing price of copper? In a nutshell, how efficient is our flue system as a value catcher?

Notes on the work as turned over to the management will be later submitted, but as a preliminary, a brief description of our flue system is illuminating. It is taken from a booklet, *Brief Description of the Washoe Smelter*, compiled by members of the A. C. M. staff: "There is an elaborate flue arrangement, especially noted for its immense size. The principal flues: viz., the blast, roaster and reverberatory, are 20 ft. wide and 15 ft. high, and are of brick and steel construction. The converter flue consists of two 7 by 7 ft. flues. The blast, roaster and converter flues connect with their respective dust chambers; the reverberatory flue with the furnaces direct (after passage of gases from each furnace through two Stirling boilers in tandem for recovery of waste heat). The flues are of the following lengths:

	Feet.
Blast flue......	1,653
Roaster flue..................	488
Converter flue..	703
Reverberatory flue...........	1,253

These flues all merge into one main flue, the rough plan of which is shown in the accompanying diagram [Fig. 2]. For the first 1,234 ft. this main flue is 60 ft. wide; side walls 20 ft. high; the bottom being excavated at

an angle of 30° from the horizontal.   The roof is of I-beam and brick arch construction.   The remaining distance to the stack is 995 ft. of 120-ft. flue (width, this last 995 ft. being practically twin flues of 60-ft. width, for further slowing velocity of gases).   This portion of the flue has a roof of No. 9 sheet steel (to enhance cooling effects by greater radiation). The stack is 300 ft. high, with an inside diameter of 30 ft.   The top of this stack is 932 ft. above the surrounding valley."   Cross-sections of the 60-ft. and the 120-ft. flues are shown in Figs. 3 and 4.

As a result of this elaborate flue system, the distances traveled by gases of various departments between furnaces and top of stack are as follows:
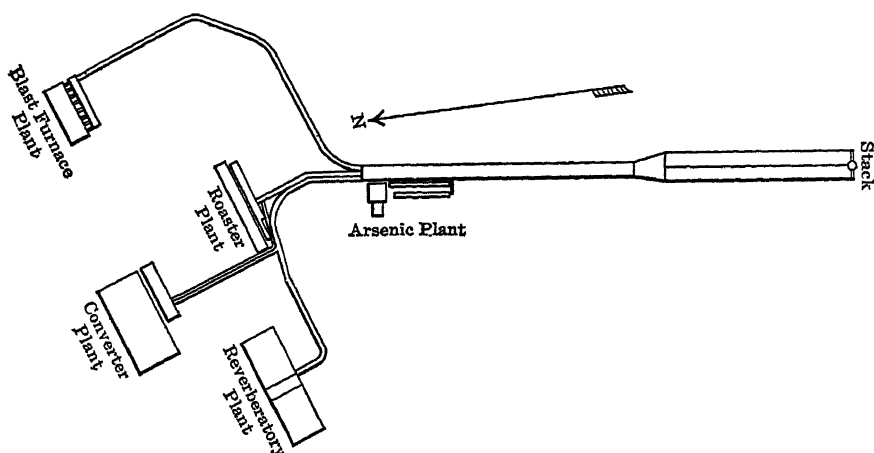
FIG. 2.—PLAN OF FLUES AT WASHOE SMELTER.

Blast gases: 1,653+1,234 at lower velocity+995 at still lower velocity and greater radiation+300 at increased velocity and reduced radiation =4,182 ft.

Roaster gases: 488+1,234+995+300 =3,017 ft.

Converter gases: 703+430 (in roaster flue)+1,234+995+300 =3,662 ft.

Reverberatory gases: 1,253+1,234+995+300 =3,782 ft.

These figures are here inserted to emphasize three effects of the Anaconda flue system, the combination of which is unique; namely: immense distance that must be traveled by dust before chance of escape is given; lowering of velocity of smoke stream of gases from all flues twice before escape; and great relative increase of radiation (with greater cooling effect) during from 20 to 25 per cent. of the distance (covered at the lowest velocity).

Its efficiency can be judged by the results contained in the notes here-
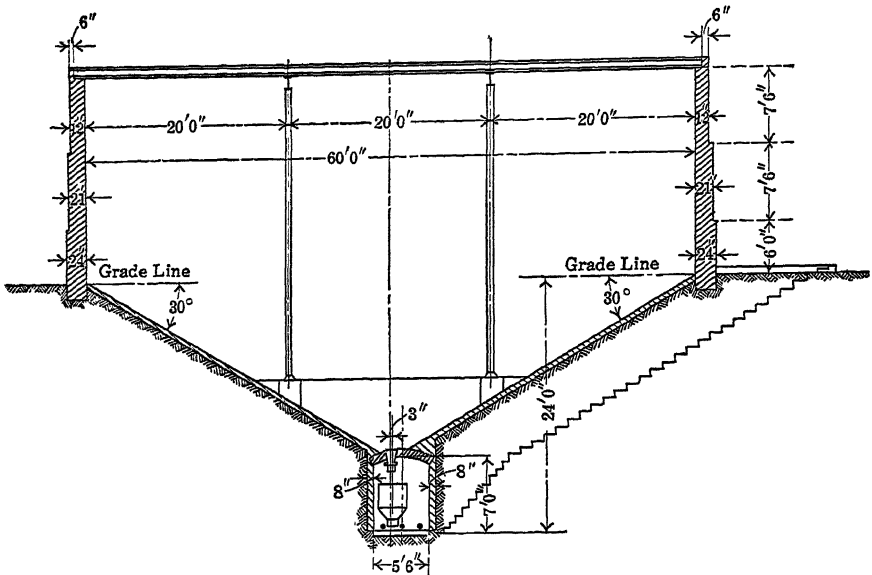
FIG. 3.—SECTION OF 60-FT. FLUE.

with submitted, covering fully methods of procedure, results obtained, and conclusions therefrom.

As the temperature of the smoke stream escaping plays a part in selecting method for analysis, that temperature is here given. The testing department records showed near base of stack for summer (1909–10) 377° F., or 192° C.; and for winter same year, 318° F., or 159° C. We ourselves checked up the temperatures 20 ft. from the base of stack in August, 1910, and obtained an average from a long series of readings of 375° F., practically that given by the testing department.
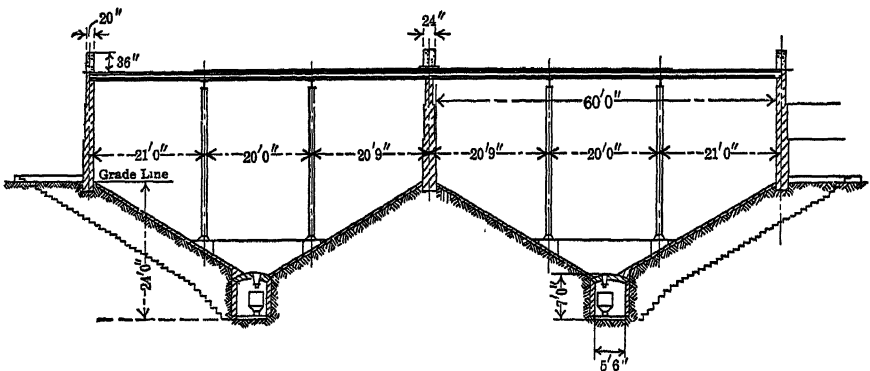
FIG. 4.—SECTION OF 120-FT. FLUE.

1. *Various Places and Methods for Determinations.*

In looking over the flue and stack preparatory to starting tests for velocities and volumes, four places for doing the experimental work were considered.

(*a*). Stack at point previously used in 1905.

(*b*). Uptakes into stack used in 1907.

(*c*). Cross-section of 120-ft. flue, 30.7 ft. below the stack, where six holes had already been cut for dust soundings.

(*d*). The cross-section of 120-ft. flue, about 70 ft. below (*c*), where ten holes had been previously cut for dust soundings.

(*a*). The opening in the stack was discarded for two reasons. First, the height from the ground (53 ft.) and the very small distance above the point at which the gases enter the stack (13 ft. above the old baffle in stack—now down) seemed to point unmistakably to varying eddies; second, the small diameter of the hole in the stack (5.75 in.) makes the insertion of a large type, well-supported Pitot tube impossible.

(*b*). The two side uptakes into the stack were discarded because the quick sweep of a large volume of the gas around the end baffle of the flue rendered the position of flow of the gas problematical, even without the chance of eddies.

(*d*). The cross-section of the 120-ft. flue about 100 ft. below the stack seemed the best point for determinations, but on taking soundings from the top of the flue, the space above the hoppers was found to be very nearly free from dust, necessitating at some places in the cross-section Pitot tubes 35 ft. long. Of course, the mechanical difficulties in using tubes of this type would require a very large set of tubes of varying lengths, with a tremendous number of readings in order to cover the open area satisfactorily.

(*c*). The cross-section of the flue 30.7 ft. below the stack wall was therefore chosen, soundings showing that but two Pitot tubes would be needed to cover the open area successfully. The bottom of the flue at this point is above the regular line of hoppers, and therefore the area as determined by soundings would be subject to no change, by removal of dust during progress of the test. Consequently a line of 18 holes (nine to each side of the flue) was cut at this cross-section. The depth of the flue at this cross-section is about 27 ft. (on the slope up to the stack wall), and a depth of dust was found by soundings (Nov. 9, 1910) of from 9.4 ft. near middle of flue to 18.45 ft. on the sides. Later soundings during progress of the test (Nov. 25 and 26, 1910) confirmed these first soundings, the greatest variation at any point being 0.35 ft.

2. Instruments Used.

The instruments used in the test are shown in Figs. 5 and 6.

Pitot Tubes.—The Pitot tubes employed in both determinations were two in number, one having an extension of 8 ft. down, the other 14 ft. They are of the same construction and of the same general type as those outlined in Fig. 5, though the impact arm is considerably larger in dimensions. They were carefully checked against each other and gave true readings.

Bristol Pyrometer.—The temperature determinations were taken with the Bristol pyrometer, which had been previously checked in several of the flues against mercury thermometers, and read correctly within very narrow limits (a few degrees under widely different temperatures). By means of this instrument, temperatures were taken at every point of Pitot readings.

Barograph and Standard Mercury Barometer.—The pressures were taken for each hole at finish of reading, by means of the barograph in the testing department, which is standardized daily by means of the United States

SECTION A–B
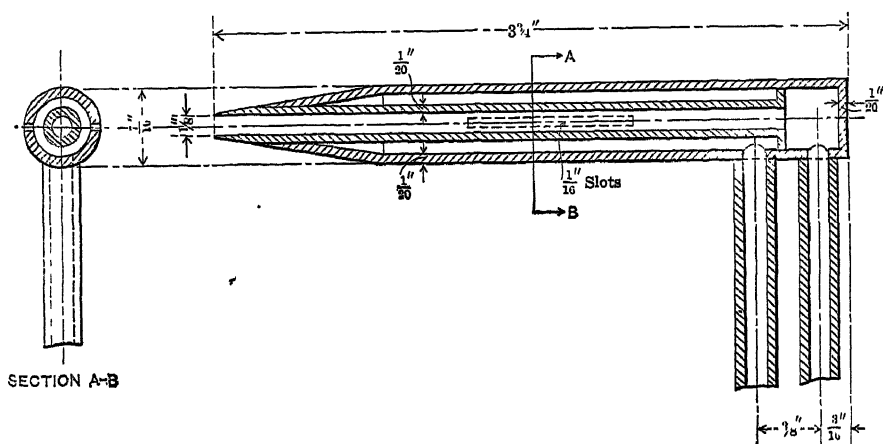
FIG. 5.—PITOT TUBE.

standard mercury barometer, also in the testing department. As a matter of fact, these pressures are slightly too high, but it was deemed more accurate to use them than to trust to an aneroid barometer at the stack, which would fluctuate too greatly during the exceedingly variable weather of the determinations.

Ellison Differential Draft Gauge.—The Ellison draft gauge proved a distinct advance over any other form of U-tube so far known for taking the Pitot readings. The gauge is made to read directly in inches of water when the Ellison gauge oil is used therein. Consequently before starting the determinations at the main flue, the two gauges (but one of which,

No. 1, was used on the main flue) were taken to the MacDougall flue and were compared with each other, using the Ellison gauge oil as a reading medium. The two gauges checked absolutely. They were then checked by means of T's inserted in the connecting tubing against each other, using gauge oil in each, and also against water in a delicate reading U-tube. Again they checked as closely as the U-tube could be read. Gauge No. 1 was then measured at various points and checked up with both oil and water, and found to read directly in inches of water when Ellison gauge
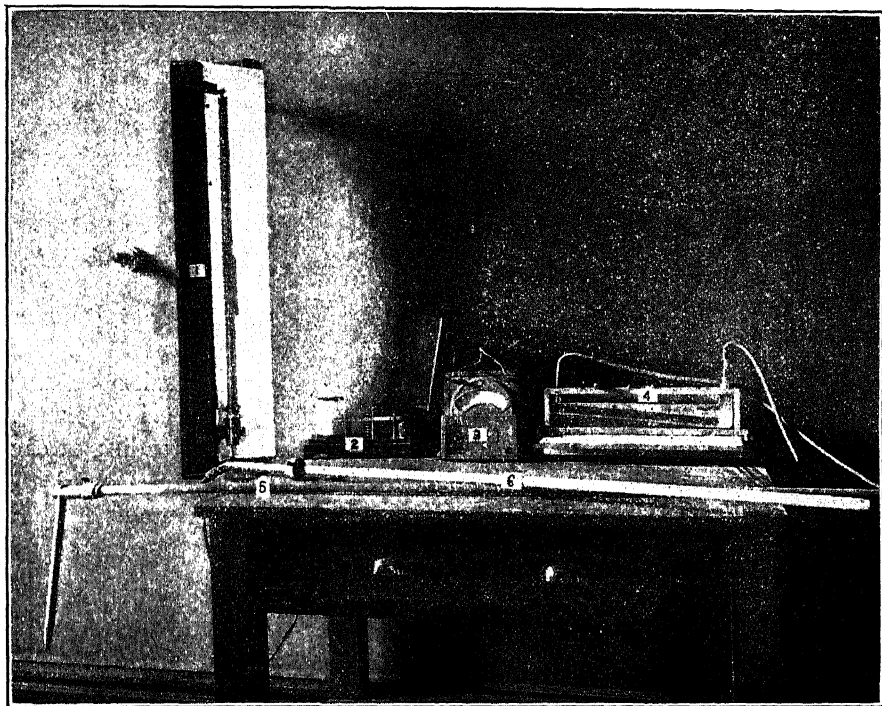
1. Government Mercury Barometer.
2. Barograph, Checked against Mercury Barometer.
3. Bristol Pyrometer, connected with 6.
4. Ellison Gauge, boxed, and on base with lag screws for leveling; connected with 5.
5. Large Size Pitot Tube.
6. Thermo-Couple in Iron Sheath.

FIG. 6.—INSTRUMENTS USED IN DUST DETERMINATION.

oil was used as a reading medium. Gauge No. 1 was then taken to the main flue and tried. The air temperature was nearly 0° C., and under these conditions the oil became too viscous for accurate work, and was therefore perforce discarded as a reading medium; absolute alcohol and petrolic ether were next in line, and the latter was adopted as a reading

medium, because of its lower specific gravity, and consequent wider range of reading velocity head minus static pressure, which remainder is small at some points in the cross-section of the main flue where work was done. Readings were tabulated directly on the scale, which had been already proved to be inches of water when using gauge oil for a reading medium. On finishing the determinations, the specific gravities of the Ellison gauge oil and the petrolic ether were determined, and each reading on the gauge converted to inches of water when using petrolic ether as a reading medium. Petrolic ether gave a specific gravity of 0.640254; Ellison oil, of 0.828625. Consequently the factor for conversion was

$$\frac{\text{Petrolic ether}}{\text{Ellison oil}} = 0.773.$$

The manufacturer's figure for specific gravity of Ellison oil is 0.834, but we found 0.829 correct. However, the small error could not be read on that part of the scale we used.

3. Procedure in Making Determinations.

In making determinations every possible precaution to guard against error was taken. The two Pitot tubes used were 8 ft. and 14 ft. long, respectively. The average height above the dust at which initial readings were taken in all 18 holes was 2 ft., succeeding readings being taken at 2-ft. distances to within 2 ft. of the top of the flue. The reading at each point is an average of three readings taken at intervals of from 1 to 2 min., so that each point reading is approximately an average of about 5 min. Each time before insertion in a hole, each Pitot tube was blown out to insure accuracy, and in each hole at a point 8 ft. below the top of the flue the Pitot tubes were checked against each other, three readings being taken with each.

To guard against the possibility of change in the specific gravity of the petrolic ether used in the Ellison gauge by absorption of dust through the Pitot tube or any other source, the petrolic ether was changed daily, the gauge being emptied every morning, the old ether thrown away, and a fresh supply used for refilling from the large, air-tight, stock bottle. Specific gravity determination of the petrolic ether was made at the end of the test, from the same stock bottle.

During readings in all holes the Pitot tubes were turned at various angles to determine the point of greatest velocity head. In all holes except two, it was found to coincide with the run of the flue. In two holes (one in each side of the 120-ft. flue) it was found necessary to turn the point of the Pitot tube toward the inside center wall of the flue, showing that gases were sweeping out toward the open stream on the outsides of the flue.

Soundings for Depth used in Area Determination.

No. of Holes Used in Making Determinations[b]	First Determination — East/West Main Flue, Nov. 9, 1910 Free Space	First Determination — Nov. 15, 1910 Free Space	First Determination Average Free Space[a]	First Determination Variations from Average	Second Determination — Nov. 15, 1910 Free Space	Second Determination — Nov. 26, 1910 Free Space	Second Determination Average Free Space[c]	Second Determination Variations from Average
1	9.50	9.85	9.68	0 18	9.85	9.50	9 68	0 18
2	11.65	11.40	11.52	0.13	11.40	11.70	11 55	0 15
3	14.20	14.50	14.35	0.15	14.50	14.40	14 45	0 05
4	15.92	16.30	16.11	0.19	16.30	16.60	16 45	0 15
5	16.95	16.80	16.88	0.08	16.80	16.70	16 75	0 05
6	17.60	17.30	17.45	0.15	17.30	17.35	17 32	0 03
7	15.15	15.00	15.08	0 08	15.00	15.20	15 10	0 10
8	11.95	11.70	11.82	0 13	11.70	11.70	11 70	0 00
9	8.70	9.00	8.85	0 15	9.00	8.80	8 90	0 10
10	8.80	8.55	8 68	0.13	8.55	8.55	8 55	0 00
11	11.10	11.10	11.10	0.00	11.10	11.00	11 05	0 05
12	13.60	13.40	13 50	0 10	13.40	13.75	13 58	0 18
13	14.63	14.55	14.59	0.04	14.55	14.20	14 38	0.18
14	14.90	14.70	14.80	0 10	14.70	14.45	14 58	0.13
15	13.90	13.65	13.78	0 13	13.65	13.60	13 62	0 03
16	13.20	13.10	13 15	0.05	13.10	12.95	13 02	0.08
17	11.00	11.15	11.08	0 08	11.15	10.80	10 98	0 18
18	8.60	8 55	8.58	0.03	8 55	8 60	8 58	0 03

Date and Place: East Main Flue, Nov. 9, 1910; East Main Flue, Nov. 15, 1910; East Main Flue, Nov. 15, 1910; East Main Flue, Nov. 26, 1910. West Main Flue, Nov. 9, 1910; West Main Flue, Nov. 15, 1910; West Main Flue, Nov. 15, 1910; West Main Flue, Nov. 26, 1910.

[a] Used as "depth" in first determination. [b] Reading east to west. [c] Used as "depth" in second determination.

The Ellison gauge was protected in every instance from the wind by a three-sided shelter (wood and corrugated iron), shown in Fig. 7, and was placed on a shaded shelf therein, to escape heat effects of flue and sun on the petrolic ether. At every point in each hole the Ellison gauge was set true by a double level, and read to zero, a drop or two of ether from a small stock bottle being occasionally added when necessary, and after taking the three readings at this point, the gauge was again checked to zero. It was found that change of position of occupants of shelter would of course throw the gauge off the level slightly, so that, after start-
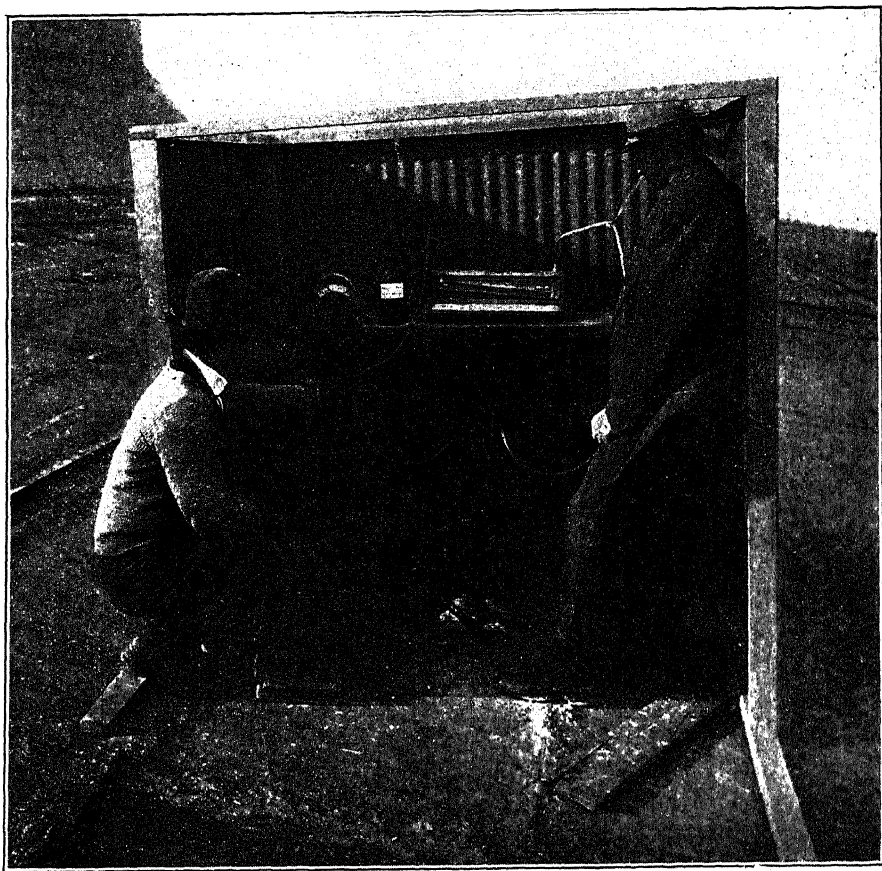
FIG. 7.—SHELTER FOR GAUGE IN DUST DETERMINATION.

ing readings at any point, absolute quiet was insisted upon. If level of gauge did not read zero, both before and after taking the readings at each point, readings recorded were thrown out, and new ones taken.

During readings, the Pitot tubes were clamped firmly in place by a

screw device, and the holes in the flue tamped with waste to eliminate any possible error from outside air currents.

In each of the two determinations, the soundings (from which the areas were figured) were made at start and end of test. Some of the 18 holes checked absolutely; others varied somewhat, the greatest variation being 0.35 ft. The averages of soundings Nos. 1 and 2 were taken in figuring the area of the first determination, and the averages of soundings Nos. 2 and 3 in figuring the area of the second determination. These soundings are appended, as they may be of interest in showing the amount of dust in flue at cross-section of soundings.

4. Calculation of Results.

Velocities.—For the calculation of velocities each hole was taken by itself in each determination, and as the points of reading were practically equidistant from each other as well as from the top and bottom of the flue, the average of the readings was taken as the average Pitot reading for each hole. The velocities were calculated by the formula $V = \sqrt{2gh}$, in which V equals velocity; g equals the acceleration due to gravity; h equals the height in feet of a column of gas equal to the displacement of the reading medium by the velocity pressure minus the static pressure. The Pitot tube is of such construction as to read the velocity pressure minus the static pressure direct. The petrolic ether displacement was first calculated to inches of water and finally to feet of water for greater ease of later calculations. In calculating, h, the specific gravity of the gas, was taken as based on former gas analyses made in previous years. The specific gravity calculated therefrom is 1.013, or a molecular weight (H = 2) for the gas of 29.2146. Or in other words, at 760 mm. pressure and 0° C. temperature, 1 liter of gas weighs 1.307689 g. or 1.308 g. (1 liter of H under same conditions weighing 0.089523 g.). Or 29.2146 g. of gas divided by 1.308 equals 22.33 liters of gas, at standard conditions. The following formula was next employed for calculating the number of liters of gas at flue conditions which are equal to 22.33 liters at standard conditions:

$$V^F = V^S \frac{(273 + t)\ 760}{273\ B} = V^S \frac{(1 + 0.00367\ t)\ 760}{B}$$

V^F equals number of liters of gas (volume) sought at flue conditions of temperature and pressure; V^S equals volume of gas at standard conditions (a constant 22.33 liters); t equals the temperature of the gas; B equals the barometric pressure. As the figures for t and B vary in the different holes, V^F was figured for each hole separately.

Areas.—In figuring areas, each side of the 120-ft. flue was taken separately, and subdivided into nine sections. The depth of the free space at each of the nine holes was determined, the figures taken being average depths of two soundings, one taken before and one after each determination of velocity. The basis of widths of section areas was the measured distances between holes and the measured distance between the outside holes and the sides of flue. The widths taken for each section area are the distances midway from hole to hole, with the exception of sections Nos. 1, 9, 10, and 18. The width of section area No. 1 is one-half the distance between holes Nos. 1 and 2, plus one-half distance to flue wall from hole No. 1. Widths of section areas Nos. 9, 10, and 18 are similarly obtained.

The remaining one-half distance between hole No. 1 and east wall of east flue was thrown out when calculating areas, for the velocity decreases in a more or less steady ratio, from the maximum at the hole to zero at the wall. This distance is of small account in figuring area, as in the four sides where it is discarded it averages about 8 ft. high (clearance above the dust), and never more than 1 ft. in width.

Width times average depth of course equals area for each section in square feet.

Pitot readings were taken at several of the 18 holes at a distance of 1 ft. from the top of the flue. In every case the Pitot reading at 1 ft. was very nearly a half of that found at 2 ft., in the same holes. This figures to a decreased velocity of approximately three-fourths as much at 1 ft. as at 2 ft.

Volume figures from areas and velocities previously calculated are certainly too high, as they assume full area of the flue and take no account of either any decrease in velocity above 2 ft. from top of flue or below 2 ft. from top of dust. Nor do they take cognizance of retarding of velocity by steel work in flue. In an attempt to ascertain the actual volume, these three things were figured on and approximated by taking off the top foot from area, assuming full velocity from 2 ft. below top of flue to 1 ft. below top; and also full velocity from 2 ft. above dust to dust; and not figuring on any retarding of velocity by steel supports of flue.

Daily Volume of Gas.—In calculating the daily amount of gases passing through the flue, each area is multiplied by its own velocity, giving cubic feet per second. Adding together the results obtained for all nine holes of each side gives the volume of gases per second in cubic feet passing each side of the flue. Multiplying by 86,400 (number of seconds in 24 hr.) gives volume of gases per 24 hr. at flue conditions of temperature and pressure.

For calculating to 0° C. temperature and 760 mm. pressure (to form

a basis for comparison with other results) the following well-known formula is employed:

$$V^s = V^F \frac{273 \text{ B}}{(273 + t) \, 760} = V^F \frac{B}{(1 + 0.00367 \, t) \, 760}$$

For B, the nine barometric readings taken during testing the nine holes to a side were averaged, and for t the temperatures taken while reading each point in the nine holes to a side were also averaged. Working out formula, this gives the daily volume of gas passing each side of the flue reduced to standard conditions of temperature and pressure.

In an attempt to approximate the daily amount of gas at all times of the year at the temperature and pressure of the atmosphere surrounding the stack, some averages of temperature and pressure for two and one-half years (February, 1906, to September, 1908) which were figured in the testing department, were taken. Figures are: temperature average, 4.6° C.; pressure average, 624.84 mm. Using the formula

$$V^a = V^s \frac{(273 + t) \, 760}{273 \text{ B}} = V^s \frac{(1 + 0.00367 \, t) \, 760}{B}$$

where V^a equals volume at atmospheric conditions; V^s equals volume at standard conditions; t = 4.6° C.; and B = 624.84 mm., the result is the average daily volume of gas after it leaves the stack, and adjusts itself to surrounding conditions. These figures are given under "Results Obtained."

Results Obtained.

		Cu Ft. per Day.
November, 1910 (full area)	No. 1	1,470,223,000
	No. 2	1,430,367,000
November, 1910 (area less 1 ft.)	No. 1	1,359,564,000
	No. 2	1,322,369,000

After calculating velocity and area and volume for each hole separately, the volume was divided by the area to obtain the average velocity (feet per second). The velocity readings for all nine holes to each side were then added together and divided by nine, to see how closely this method would check the other for velocity feet per second per side of flue. Figures follow:

	av ÷ a = Average Velocity Feet per Sec.	9v ÷ 9 = Average Velocity Feet per Sec.
East Side main flue:		
First determination	21.19	21.07
Second determination	21.94	21.70
West Side main flue:		
First determination	22.01	21.83
Second determination	21.26	21.12

The maximum velocity was found at hole No. 2 on the east side in both determinations; and at hole No. 17 on the west side in both determinations; while in both determinations, the minimum velocity was at the holes nearest the center dividing wall of the 120-ft. flue—hole No. 9 on the east and hole No. 10 on the west, all holes being numbered from east to west across the entire 120-ft. flue.

	East Side—Main Flue Feet per Second	West Side—Main Flue Feet per Second
Maximum Velocity:		
First determination	Hole No. 2—28 90	Hole No. 17—29 72
Second determination	Hole No. 2—31.27	Hole No. 17—29 23
Minimum Velocity:		
First determination	Hole No. 9—11 84	Hole No. 10—12 37
Second determination.	Hole No. 9—12 42	Hole No. 10—12 30

Over 700 Pitot readings were taken in the two determinations.

The volumes of gases emanating from stack were, for 24 hours:

Cu. Ft

Flue conditions   . . .   .   169° C. and 622.9 mm. = 2,649,530.000
Standard conditions .   . . .   0° C. and 760   mm. = 1,340,960,000

The average velocity in one 120-ft. flue was (assuming 80 per cent. clear space) 12 ft. per second.

Plant Operations During Tests.—During the first determination, the following units of the plant were in operation: Five blast-furnace units; seven reverberatory furnaces; 37.46 MacDougall furnaces; eight converters. During the second determination, blast furnaces and converters were the same, while six reverberatory and 29.43 MacDougall furnaces completed the operation. ·

Summarized on a basis of maximum capacity:

	First Determination	Second Determination
Blast furnaces.	5/7	5/7
Reverberatory furnaces...	7/8	6/8
MacDougall furnaces[a]	37.46/64	29.43/64
Converters	8/13	8/13

[a] MacDougall furnaces given on basis of 64 furnaces—the *full* equipment.

DUST DETERMINATIONS AT STACK.

1. *Place, Apparatus, and Methods of Sampling.*

Place.—The place chosen for sampling the dust and fume emanating from the Washoe Smelter flue was the same as that selected for velocity determinations, a line across the large 120-ft. flue, 30.7 ft. below the base

of the stack wall.   Fig. 8 is a sketch showing the dust-catching apparatus. This place was chosen in preference to the two others considered (stack, and arches into the stack) largely for the same reason for which it was given the preference in former velocity work: namely, to escape swirling eddies, which has been found to occur largely in both places aforementioned. Then, too, the apparatus employed was of such a nature as to render the use of the sampling station in the stack, at a considerable height above the ground, impossible.   Of course, it must not be forgotten that some small percentage of the dust (which passes this line 30.7 ft. below the stack wall) is probably precipitated during its later passage into the arches and up the stack.   But that a very small percentage, even of this precipitated dust and fume, falls *inside* the stack is shown by the relatively long time required for assembling the dust in the hoppers at the base of the stack.   Reasons why so little dust and fume are caught after passing the sampling point are two-fold.   The first (based on velocities) is that the area being so soon constricted after the passage of the sampling point, there is a quick corresponding jump in velocity from approximately 21 ft. per second (average of flue) at sampling line to about 46 ft. per second in the stack, which would tend to overcome any great tendency of dust to settle by precipitation caused by greater friction on walls of stack over walls of flue, and, consequently, the greater part of such dust and fume, *if precipitated*, would largely be swept out at top of stack. Then, too, the time in flue and stack after passing sampling line is so short (about 8 sec.) that precipitation after passing stations must be slight indeed.

The second reason is arrived at by analysis of dusts.   The insoluble is so low as to stamp any further dust settling after passing sampling stations as slight, and the conclusion is forced that the flue, *as now constituted*, possesses a remarkably high efficiency as a *dust* settler, nearly everything now passing the sampling stations 30.7 ft. below the stack wall being *"fume"* so called, and not precipitable under the slightly changed conditions beyond the sampling stations, neither in the flue itself nor yet in the stack.   So much for possible criticisms as to the possibility of dust obtained at sampling stations 30.7 ft. below stack wall being larger in any great degree than the emanations from top of stack. Of course, there is some later precipitation, else there could be no dust in hoppers under the stack and above sampling stations, but, for all reasons just mentioned, we believe it to be far below 1 per cent., which is certainly within the errors of sampling.

Apparatus and Methods of Sampling.—In 1905, dust determinations were attempted in the stack, measuring the volume of gas (containing the dust obtained) by means of water aspiration, using a meter for gas readings.   These samples averaged something like 70 cu. ft. of gas
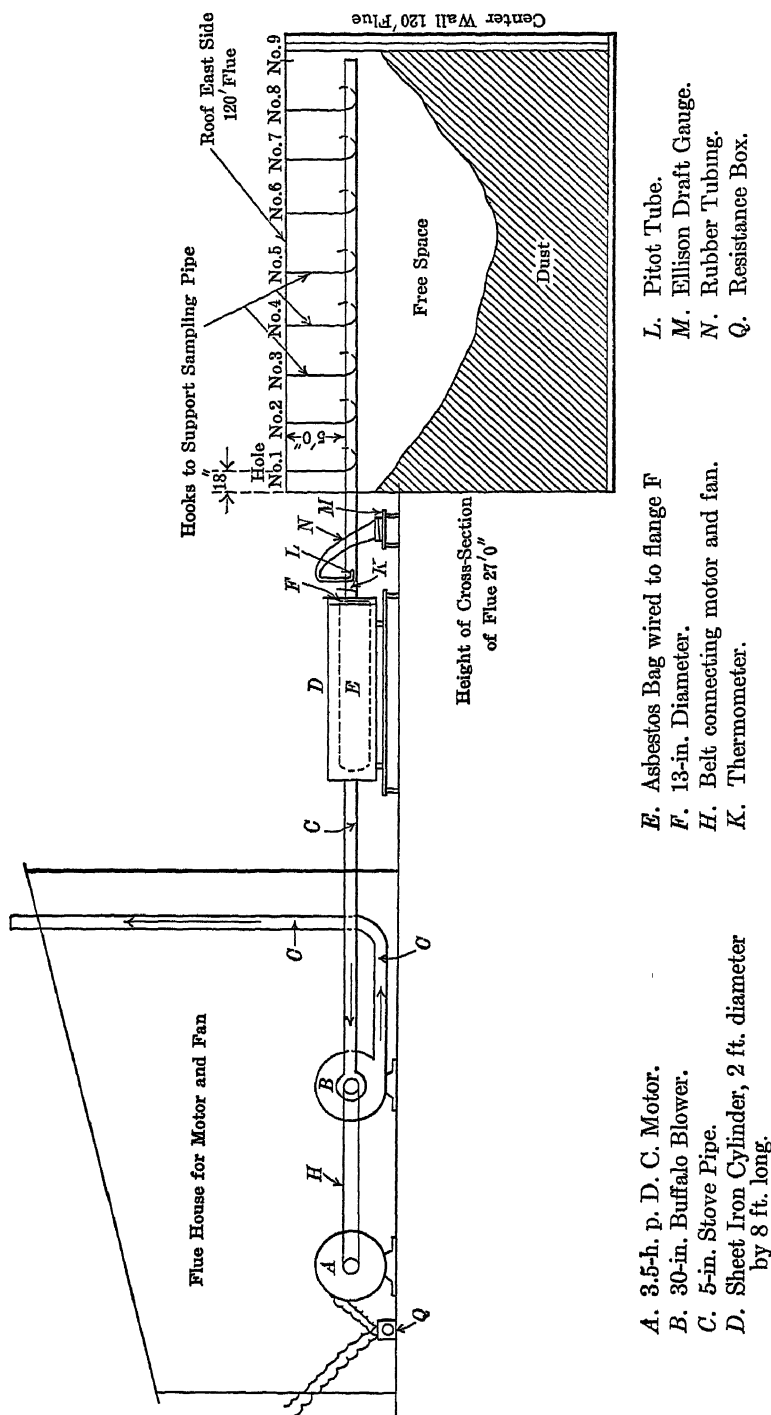
FIG. 8.—DUST-CATCHING APPARATUS AND CROSS-SECTION OF EAST HALF OF 120-FT. FLUE, SHOWING OPEN AREA AT TIME OF DUST DETERMINATION

A. 3.5-h. p. D. C. Motor.
B. 30-in. Buffalo Blower.
C. 5-in. Stove Pipe.
D. Sheet Iron Cylinder, 2 ft. diameter by 8 ft. long.

E. Asbestos Bag wired to flange F
F. 13-in. Diameter.
H. Belt connecting motor and fan.
K. Thermometer.

L. Pitot Tube.
M. Ellison Draft Gauge.
N. Rubber Tubing.
Q. Resistance Box.

(standard conditions) and a fraction of a gram of dust. Such methods might be termed "chemical" from the smallness of measures obtained, and while perhaps accurate for the short time employed, are yet questionable when applied to a volume like 1,450,000,000 cu. ft. per day (standard conditions) and nearly double that amount at flue conditions. We soon gave up all such "chemical" methods, and turned perforce to some determinations made by the testing department in 1907. After a thorough trial, we were convinced that this method (with some variations) would give the best possible results. In brief, the gases were drawn from the flue by means of a 30-in. rotary blower fan, driven by a variable speed 3.5-h.p. electric motor. The dust and fume contained in the gases were filtered out by means of a very closely woven thick "blanket" asbestos bag, suspended in a large air-tight drum; and the velocity (and consequently the volume) of gases passed was obtained by readings with small Pitot tubes and thermometers placed in the sampling tube between the flue and the drum.

The filtering medium was tried out thoroughly, and proved much more efficient than the regular woolen bag of the bag house; in fact, efficiency is absolute as far as dust is concerned, and nearly the same for fume. Of course, the efficiency of the filter is aided by two things: lowering of temperature due to radiation from large drum surface, and great decrease in velocity in the bag over the velocity in the sampling pipe. The temperature in the bag is still high enough to be above the condensation point of the water vapor but below that of As_2O_3.

Variations from previous methods were as follows: In the first place, they made use of the arches of the stack, which we discarded at once because of the presence of eddies. Then, too, they made use of a very small sampling pipe in order that they might with a *fixed* speed motor obtain the same velocity in sampling pipe as in the flue. This necessitated taking Pitot readings either in the very small sampling tube, where the friction on the sides would be relatively enormous, and correct readings very hard to obtain; or else in the large 5.5-in. pipe between the drum and the fan, which would include any leakage around the removable head of the drum, making gas readings too high and dust readings consequently too low. Our testing engineer insisted that dust must be drawn from the flue at a velocity at least as high as that within the flue, otherwise the sample might contain less dust and fume than present in the smoke stream. He frankly acknowledged no proof in support of his opinion, so we tried it out. We made five runs in the same place, three with 5.5-in. sampling pipe, with an average velocity of about 20 ft. per second in the sampling pipe, and two with 3-in. sampling pipe and an average velocity therein of approximately 50 ft. per second. In all five instances the flue velocity was about 28 ft. per second. We experienced

greater fluctuations in Pitot readings with the 3-in. pipe, and consequently greater probabilities of inaccuracy of reading, but when we figured the dusts there was slight variation in them, the said slight variation being easily ascribable to difference in dust content on different days.

Consequently, we adopted the large 5.5-in. pipe as a sampling tube for our determinations, and took the readings between the flue and the drum, which eliminated all chance of reading in any leakage of air.

Preliminary runs were made to determine the life of the bag, and the filter was found efficient up to 20 hr. Judged by the appearance of the bag at the end of that time, its lifetime is probably about 30 hr. in the dust stream of the drum bag holder, though it would probably be less than that if used in the flue itself at slightly higher temperature.

Runs of from 100 to 120 min. at various places in the flue were found to check as closely as longer runs, and were therefore adopted as time for regular determinations. As compared with 1905 determinations, with 70 cu. ft. (standard conditions) and trace of dust, we obtained about 10,000 cu. ft. (standard conditions) and from 400 to 500 g. of dust. We experienced mechanical difficulties with the sampling pipe, and finally adopted the scheme of first running in the full 60 ft. of pipe, strongly riveted, about 58 ft. into the flue from each side, supporting it with hooks hung from the steel roof of the flue through small holes pierced therein, and pulling out and cutting off the pipe at lengths corresponding to the stations where velocity measurements had previously been taken. This gave stations on both east and west sides of flue, with a total of 18 for the entire 120-ft. flue. For these 18 stations, 28 samples were taken, besides a large number of preliminary runs in trying out the apparatus.

For the east side, all samples were taken in a line across about 5 ft. below the steel roof, while for the west side, 6 ft. below was approximated.

During each determination Pitot readings were taken every 2 min. during the entire run, and the 2-min. readings were all taken at each station, in the center of the 5.5-in. pipe. This pipe was sectioned into six areas, and in each station, with its different length of sampling pipe, the readings in the center of the pipe were corrected by a factor obtained by repeatedly reading the Pitot tube in all six areas, and applying the average reading (or rather the square roots of same, averaged and squared) to the center reading.

As the velocities vary as the square roots of the Pitot readings, the square roots of the 2-min. readings were averaged and squared to obtain the average Pitot reading for the entire run. As the temperatures were of slight variation, they were also averaged for the entire run.

As the dust on each side approached within about 8 ft. of the steel roof of the flue, the necessity of fighting shy of settled dust rendered it impossible to sample horizontally at a lower depth than 6 ft., though the

centers of both sides of the flue showed a clear space of from 16 to 17 ft. below the steel roof. But sampling perpendicularly is open to the objection of low results by loss of some of the dust that strikes and tends to precipitate on sides of sampling tube, later falling back. Dust in the gas might very well vary at a distance of 5 ft. to 16 ft. below the flue roof, because of difference in velocity of gases at the two points; but it is probable that fume does not, for we found pretty nearly the same amount of dust (and fume) all the way across the flue at stations with velocities ranging from 11 to 31 ft. per second, and it is reasonable to suppose that if no great variation occurs horizontally within such a range of velocities perpendicular variations are also slight.

It was found best to cement the bag by a preliminary run before each determination. The dust thus obtained was left in the bag and weighed *in toto* (bag and dust) and after the regular run the bag was again taken out and weighed with the contained dust. Gain in weight equals dust obtained during determination. The bag was then shaken out thoroughly and the dust bottled for screening (to take out asbestos particles) and analysis.

Before putting in the bag and drum, a preliminary run of a few minutes at high speed with rapping and shaking of pipe was made, and the pipe was shaken and rapped at intervals of the regular run, and finally just before the finish.

The Pitot tube was read, as in the regular velocity determinations, by means of the Ellison differential gauge, containing petrolic ether.

To sum up, low dust results might be obtained by reading in air leaks or by taking the velocity readings in the center of the pipe without correction for areas of pipe where velocity is lower because of friction. Both of these cause low results by giving too high a volume of gas for the dust obtained. Low results again may be obtained if the small amount of dust collecting in sampling pipe is not drawn out into the bag by increasing the velocity at end of run, and shaking and rapping the pipe. All of these errors were carefully guarded against. High dust results might occur from taking the Pitot readings in an outside area of the sampling tube, without a correction for the increased velocity in the center.

Either high or low dust results might occur from incorrect gauges for reading, or imperfect determinations of the specific gravity of the petrolic ether used as a medium. So far as known, every possible error has been carefully eliminated from the work.

2. Results Obtained.

The figures for the dusts here submitted were obtained by two methods of figuring: First and simplest, by taking the arithmetical average of the dusts obtained at all stations, based on two standards (for which see

previous report of velocity determinations). Assuming the figure 1,450,-000,000 cu. ft. for the daily volume of gas at standard conditions gives the first figures; while the second (more nearly correct—for reasons see previous report of velocity determinations) are based on a daily discharge of 1,341,000,000 cu. ft. at standard conditions.

After figuring the arithmetical average, the dust figures were also obtained by a more correct method of calculation, each dust being calculated by its own area and velocity, using the figures obtained from previous velocity report. These results checked the arithmetical average within much less than 1 per cent.

Dusts Obtained in Main Flue.
East Side (Arithmetical Average).

DATE OF SAMPLE, 1911	Station No	Velocity in Pipe Feet per Second	Dust Tons per Day Based on 1,450,000,000 Cu. Ft.	Dust Tons per Day Based on 1,341,000,000 Cu. Ft.
Jan. 16...................	1	19.325	87.000	80.460
Jan. 20............. .	1	15.186	79.469	73.495
Jan. 21...............	1	50.560	85.672	79.232
Jan. 30 ...:....... . . .	2	21.176	85.927	79.458
Jan. 25	3	17.678	83.239	76.982
Jan. 26....... . . .	4	18.722	83.784	77.486
Feb. 17..........	4	19 147	84.482	78.131
Feb. 14.......	5	19.823	81 111	75.014
Feb. 13........ .	6	19.138	88.276	81.641
Feb. 11.......	7	19 138	82 927	76 694
Feb. 7........	8	17.197	78.828	72 904
Feb. 6.......... ..	8	20.200	79.594	73.609
Feb. 4..............	8	18.170	82.675	76.461
Feb. 4.............	9	17 657	84.151	77.826
Feb. 1..........	9	17 818	83.431	77.160
Arithmetical average.. .	.	....	83.371	77.104

West Side (Arithmetical Average).

Mar. 1.	10	17.333	72 68	66.43
Mar. 2......... ...	10	18.256	75.02	69 38
Mar. 2.....	11	18.292	73.06	67.57
Mar. 3.	12	18.278	80.01	73.99
Mar. 4......	13	18.606	93 45	86.44
Mar. 6....	13	17.695	85.80	79.35
Mar. 6..	14	19.247	93.20	86.20
Mar. 7.	14	18.018	95.91	88.69
Mar. 7.... .. ..:	15	19.080	91.51	84.63
Mar. 9.....	16	19.675	77.03	71.23
Mar. 10...	16	17.251	77.17	71.37
Mar. 10..........	17	18.362	81.79	75.65
Mar. 11..	18	16.416	78.54	72.63
Arithmetical average. ..	.... ..	....	82.713	76.428
Arithmetical average (both sides).............	....	....	83.066	76.790

Figuring the dust by the roundabout but exact way aforementioned (ascribing to each sample its own flue velocity and area for each separate sample and station) gives correct figures as follows: Using 1,450,000,000 cu. ft. as daily discharge at standard conditions from both sides of flue, gives 83.015 tons of dust per day against 83.066 as the arithmetical average; while employing the figure that we believe to be more nearly correct, viz., 1,341,000,000 cu. ft. per day, gives 76.777 tons per day against 76.790 arithmetical average.

In the east side of 120-ft. flue there is small variation in dusts at different stations; while in the west side the greater variation is probably due to greater velocity in the west flue. There is more nearly a swirling eddy at stations 13, 14, and 15 (west side) than at stations 5, 4, and 3 (east side), the velocity in the west side being slightly higher, with greater, bounding back from baffle 30 ft. above stations. The greater amount of fume on the east side is probably due to imperfect admixture of gas and fume from the sectional flues in the 60-ft. flue before entering the 120-ft. flue. Slight differences in previous gas analyses showed similar evidence.

Analysis of Dusts.—Eighteen samples were made up for analysis, all being screened through 40 mesh for removal of asbestos from bags. They represent the dusts at the eighteen different stations of sampling

East Side of Flue.	*West Side of Flue.*
Nos. 1 and 8. Composite 3 determinations	Nos. 10, 13, 14, and 16. Composite 2 determinations
Nos. 2, 4, and 9. Composite 2 determinations	Nos. 11, 12, 15, 17, and 18. One determination only
Nos. 3, 5, 6, and 7. One determination only	

These 18 samples were analyzed, with the following results:

NUMBER	Ag Oz. per Ton	Au Oz. per Ton	Cu Per Cent.	As$_2$O$_3$ Per Cent.
1	3.8	0 005	1 05	40.3
2	4.0	0.005	0 92	40.6
3	3.9	0 005	1.03	40.7
4	3.4	0.005	0.96	39 9
5	4.1	0 006	0.95	31.0
6	4.2	0.006	0.90	32.8
7	4 4	0 006	1.15	31.1
8	4.3	0.006	0.92	31.6
9	4.3	0.006	1 10	35.7
10	4.9	0.006	0.95	29.7
11	5.5	0.007	0.82	33.0
12	5.0	0.006	1.13	30 8
13	4.5	0.006	1.01	33.2
14	4.7	0.006	1.02	30.8
15	4.7	0.006	0.91	39.4
16	4.1	0.005	0.82	32.2
17	4.8	0.006	0.93	32.4
18	5.3	0.006	1.18	31.3
Average.......	4.44	0.0058	0.986	34.2

The methods used for copper (color and electrolytic) gave results far too high, as shown by our own later analysis. A composite sample of 10 g. of each of the 18 samples was made up and thoroughly mixed, and run in the fall of 1911. Results are herewith submitted:

Analysis of Main Flue Dust Passing Base of Stack.

	Per Cent.	Ounces
SiO_2	4 19	
Insol	5 68	
Al_2O_3	3 14	
Fe_2O_3	2.78	
CaO	0 50	
MgO	0 69	
Total S	10 05	
Total SO_3	23.25	
Free SO_3	3 60	
Copper colorimetric	0 91	
Copper electrolytic	0 84	
Copper iodide (Sept. 6, 1912)	0 85	
Gold		0 005
Silver	0 02	4 9
Zinc	9 50	
As_2O_3	34.34	
Pb	10.80	
Sb_2O_3	1 17	
Bi_2O_3	1.15	
Na_2O	0.49	
K_2O	0 42	
Carbon	0.8	
Tellurium	0.28	
Selenium	Nil	

Gold and Silver by Hunter. Tellurium and Selenium by Ware. Analysis by Dunn.

Sample taken for analysis was a mixed one of the dusts taken from 18 stations across the 120-ft. flue, 30.7 ft. below the base of stack. The dust was drawn out by means of a motor and suction fan, and was caught in asbestos bags. Evidences of the bags are seen in the analysis, particularly in SiO_2, Insol., MgO, K_2O and Na_2O.

Flue Dust Analysis—Totaled.

	Per Cent.
SiO_2	4.19
Al_2O_3	3.14
Fe_2O_3	2.78
CaO	0.50
MgO	0 69
Free SO_3	3.60
Combined SO_3	19.65
Sulphur free and as sulphide	1.88
CuO electrolytic	1.05
ZnO	11.84
As_2O_3	34.34

		Per Cent.
PbO	. ...	11.63
Sl₂O₃	. . . .	1 17
Bi₂O₃	. .. .	1.15
Na₂O.	. . . .	0.49
K₂O.	. .. .	0 42
Carbon	. . ..	0.8
TeO₂	. . ..	0 35
Ag₂O	. ...	0 02
		99 69

Fine asbestos is plainly present (from the bags used for catching dust), vitiating the analysis somewhat.

Six months standing (between sampling and analysis) may also have increased the combined SO_3 at the expense of the free SO_3.

Tellurium may be present as TeO_3 in combination as tellurate, which would increase total slightly.

Our thanks are due to H. S. Ware, S. D. Hunter, and A. Austin (of the Washoe Laboratory force of 1911) for various analyses; and to J. O. Elton, H. H. Goe, and P. A. Haines (of the testing department and laboratory force of that time) for valued assistance in work on the flues.

As to commercial side of dust recovery, we append portion of report of Apr. 8, 1911, as follows:

"A partial analysis of the dusts taken during the determinations at the stack gives the following values (arithmetical averages being employed):

	Ag Ounces	Au Ounces	Cu Per Cent.	As₂O₃ Per Cent.
East side flue...	4.04	0.0056	0 998	36 0
West side flue....	4 83	0.0060	0.974	32.5
Average, two sides........	4.44	0.0058	0 986	34.2

"Expressed in values per ton of dust the average figures give about $2.25 Ag; $0.12 Au; $2.40 Cu; or about $4.75 per ton for what one might call 'fixed values.' Figured to 77 tons per day (daily dust and fume emanating from stack) shows a daily loss from stack of $365.75, a figure which would hardly pay for smelting the refractory material, should it be recovered by any process. Add to smelting cost the initial cost of installation of dust and fume device, and the daily cost of labor and power for same, and the recovery of our solid stack emanations becomes

a matter outside the realms of commercial processes, and means a big charge against smoke expense.

"Nor will the recovery of the arsenic aid the matter materially, for this substance is what might be called a 'variable value.' At present the market is 'dull and hard to move' at \$1.875 per 100 lb. white arsenic (*Engineering and Mining Journal*, Apr. 1, 1911). While this gives a figure of \$37.50 per ton for As_2O_3 (with about 26 tons daily, or \$975 daily value), yet the value of the As_2O_3 is a far different thing from the values of the other commercial elements. Its recovery means special treatment at the arsenic plant, with probable installation of bag house at the end of the kitchens before re-entrance of waste gases into main flue. And its recovery and entrance into an already dull and sagging market means breaking of present prices to a vanishing point, or at least to a point where the price would not pay for delivery. Consequently, that \$975 per day loss of arsenic is a problematical loss; that is, one where recovery might mean an additional loss, and an added charge to smoke expense. Commercially, there is nothing in the line of dust and fume now emanating from the stack of the Washoe Smelter which would pay for recovery, treatment, and delivery of product.

"The efficiency of present flue system is shown by a screening test of the material recovered in our experiments. All of the 18 samples employed in analysis were screened through 40 mesh to eliminate (in the oversize) the asbestos from the bags used as filters. Within 10 min. the free SO_3 and H_2SO_4 in all samples chokes a screen of any size, but by brushing the dusts with a large camel's hair brush, everything except the asbestos was put through the 40 mesh. In some cases there was a trace remainder from the 400–500 g. of sample. A mixed sample of 200 g. was then made up from approximately equal amounts of all samples used in the analysis, and by means of the camel's hair brush, 100 g. of this composite was put through a 200-mesh screen. The oversize was astonishingly small—0.046 g., or about 0.05 per cent. If we add to this the trace obtained in some instances as oversize 40 mesh in the first screening the percentage of what one might call dust still remains below 0.1 per cent., giving about 99.9 per cent. fume, which would require widely different methods (from those of our present flue system) for recovery.

"The oversize on 200 mesh was too small for analysis, but a powerful glass betrayed the presence of siliceous particles and pyrite. A magnet showed a considerable percentage of magnetic material, probably roasted pyrite."

Later analyses checked the silver, gold, and arsenic figures, but showed the actual amount of copper present as shown by first results to be too high. The discovery of relatively high amounts of Zn and Pb aids conditions somewhat, though they are in such a finely divided state as to

require special methods of treatment for their smelting, if the stack loss could be recovered.

There can be no question that the installation of any device for catching the *fume* (*dust* being at present well settled) on the enormous scale required by the daily volume of stack emanation is not a commercial proposition; a large part of the cost of such installation and daily upkeep of same, being of necessity an additional charge against present smelting cost.

References on Pitot Tube and Calculations.

1.   Jager and Westby: Determination of the Velocity of Gas with the Pitot Tube, *Engineering and Mining Journal*, Vol. LXXXVIII, p. 468.

2.   C. E. McQuigg: Pitot Tube in Gas Measurement, *Engineering and Mining Journal*, Vol. CXV, p. 649.

Arsenic Trioxide from Flue Dust.

BY JAMES O. ELTON, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

THIS paper covers, besides laboratory work, a study of actual operation at the Washoe Smelter over a considerable period of time, together with the results of a visit to the Midvale plant of the United States Smelting & Refining Co., near Salt Lake. For the latter privilege, as well as permission to publish the resulting data herein given, I am indebted to George W. Heintz, General Manager of the Midvale plant, and here take pleasure in expressing my appreciation for the many courtesies extended to me by Mr. Heintz and members of the technical staff at the smelter.

Almost all of the world's supply of arsenic is recovered as a by-product in the smelting of other metals. It is seen on the market in various forms, the most common being arsenic trioxide, the "white arsenic" of commerce, which is sold as a heavy white powder, or in white, glassy, translucent masses. The trioxide is used in the manufacture of pigment, glass, and insecticides. About 50 per cent. of the domestic consumption is supplied by imports.

Statistics of Arsenic Trioxide in United States.[1]

Year.	Production. Tons	Consumption. Tons.	Price Per Pound. Cents.
1907, ... 1,010		5,471	5
1908, ... 1,302		6,098	$3\frac{3}{8}$
1909, ... 1,007		4,600	$2\frac{7}{8}$
1910, ... 1,526		5,428	$2\frac{1}{4}$
1911, ... 3,081		6,539	2
1912,[2] 2,926			Closed at $4\frac{7}{8}$

Arsenic occurs in combination with sulphur and the metals in a large number of minerals. It is present in small amounts in many ores. In the roasting and smelting furnaces many of its compounds break down, liberating arsenic trioxide, which is volatilized and carried by the gases into the flue system, where a part is condensed and settles with the flue dust.

[1] *Mineral Industry*, vol. xx. (1911.)
[2] *Engineering and Mining Journal*, vol. xcv., p. 96 (Jan. 11, 1913).

Flue dust is the name applied by metallurgists to the solid material caught in smelter flue systems. It includes both dust and fume. Dust consists of fine particles of the charge, or products of the charge, blown or carried by the gas current into the flue system. All smelter charges contain certain substances which volatilize wholly or in part when subjected to high heat in the reducing or oxidizing zones of the furnaces. Upon cooling, these vapors condense, forming minute particles known as fume. Dust can be settled without lowering the temperature, while fume must be condensed before settlement is possible.

The Anaconda metallurgists recognized the fact that in copper smelting the copper values are nearly all contained in the dust, and designed their flue system to catch the maximum amount of dust. Large dust chambers near the furnaces settle the coarse dust. The finer dust, along with the fume, is carried by the gases through departmental flues to the main flue system. As the temperature decreases fume condenses, and a part settles with the dust. The amount settled is dependent upon the decrease in temperature and velocity. The main flue is 2,230 ft. long and discharges into a brick stack 300 ft. high and 30 ft. in inside diameter. The first 1,230 ft. of the main flue is 60 ft. wide and 30 ft. deep, with a center line of hoppers spaced 5 ft. apart, for the removal of dust. The remainder of the flue is 120 ft. wide and is constructed as two parallel 60-ft. flues with a common center wall. There is a line of hoppers under each section. The dust from the first 195 hoppers is sent to the reverberatories and is smelted direct; the last 115 hoppers supply the feed for the arsenic plant.

The general practice has been to save the metal values and allow the fume to escape, unless it contained other valuable constituents, or damage to animal and vegetable life compelled its removal.

The separation of fume from gas may be accomplished by filtering through bags, settling in large chambers hung with wires, or precipitating by electrostatic apparatus. When the fume is settled wholly or in part, it must be disposed of, or the accumulation becomes a burden on the system, if not stopping it entirely. At Anaconda, the flue dust from the last 115 hoppers of the main flue has been treated to recover As_2O_3 since 1903. The production of arsenic has been small, and costs have been high. In January, 1913, an investigation[3] was begun with a view to improving recovery and increasing the tonnage treated.

Arsenic Recovery.

Arsenic trioxide is recovered from fume or flue dust by roasting or distilling off the volatile As_2O_3 and collecting the fumes in condensing chambers. The settled product, called "crude arsenic," is redistilled to further purify it. This refined product often contains 99.97 per cent. of As_2O_3. It is pulverized and barreled for market.

Available Arsenic.

A study of the arsenic plant records showed very erratic recoveries, varying between 30 and 70 per cent. Experiments were undertaken to locate the trouble and to determine the proper conditions for maximum "burnout."

A coal-fired muffle assay furnace was selected for the roasting experiments. The temperature was measured with a Bristol pyrometer and thermo-couple. Ten-gram samples of the dust contained in 6-in. roasting dishes were roasted at different temperatures, care being taken to maintain as nearly as possible the same heat throughout the roasting.

Two representative samples were selected, one a composite made up from a number of "hopper dusts." The other was a sample taken from Hopper 195. (The arsenic plant feed comes from hoppers 195 to 315.)

When the desired temperature was obtained in the furnace, five samples of each dust were placed in the muffle in such a way that the corresponding number of each sample would be subjected to the same furnace variation. Every 5 min. the temperature was read and recorded. At the end of 10, 20, 30, 40 and 60 min. a sample of each dust was removed from the furnace, cooled, and weighed. Later these residues were analyzed for As_2O_3. The amount of As_2O_3 volatilized was calculated, and curves were constructed showing the percentage decrease in weight due to As_2O_3 and weight burnout (A^1 to A^6, Figs. 1 and 2). The above procedure was followed with roasts made at temperatures ranging from 800° F. to 1,600° F. Using the above results, a composite curve was also constructed, showing the relative "burnout" of total weight and As_2O_3.

To show the effect of a gradually increasing temperature, the samples were placed in the furnace when cooled to 600° F. and the temperature gradually raised to 1,600° F. At the regular intervals, samples were removed as before, cooled, weighed, and assayed. (Roast A^7, Fig. 2.) Sulphur and zinc "burnout" curves were constructed, showing the elimination of these elements.

The legend on the plots gives the results obtained.

At the higher temperatures, there was a difference of nearly 200° in different parts of the muffle. This explains the seeming variation in Roasts A^4, A^5 and A^6. In all roasts except A^1 and A^7, the "burnout" of the volatile As_2O_3 was complete at the end of 10 min.
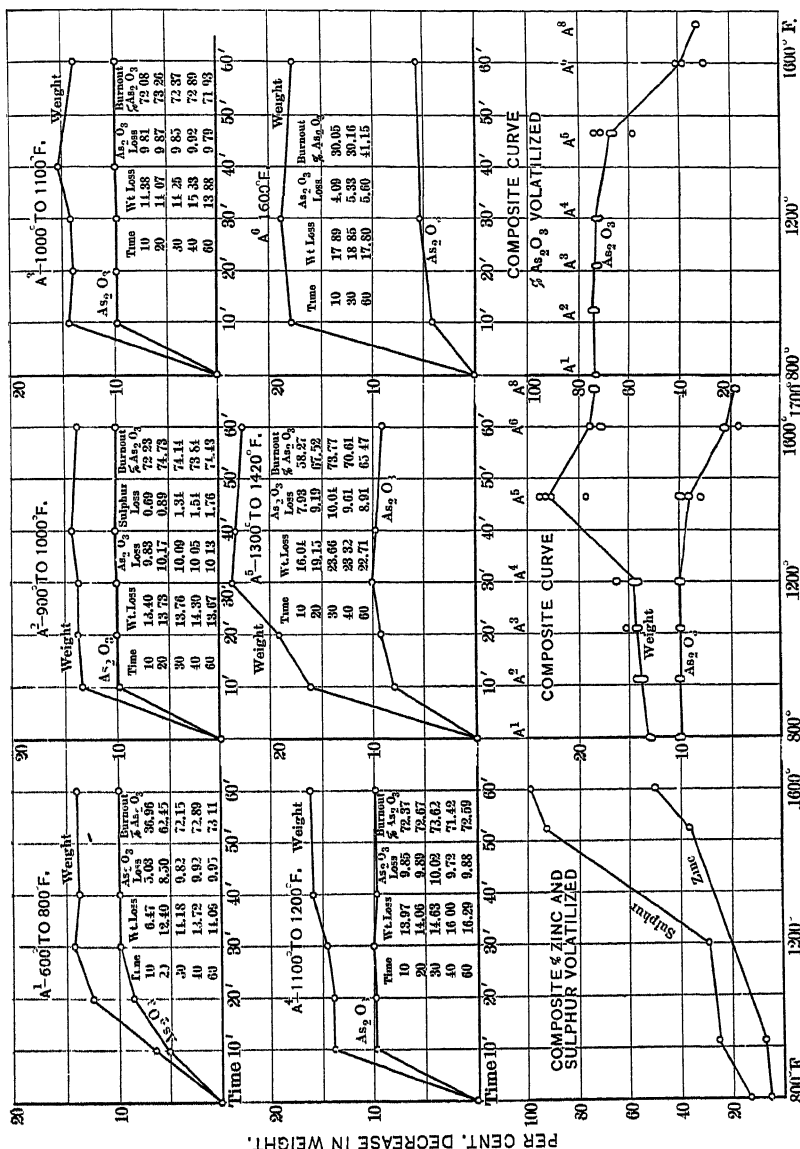
FIG. 1.—MUFFLE ROAST ON FLUE DUST. SHOWING PER CENT. OF TOTAL WEIGHT VOLATILIZED, PER CENT. OF DECREASE IN WEIGHT DUE TO As_2O_3 BURNED OUT, WITH TIME AND TEMPERATURE VARIATION, ON COMPOSITE SAMPLE.

A^1—600° TO 800° F.

Time	Wt. Loss	As_2O_3 Loss	Burnout % As_2O_3
10	6.47	3.08	36.96
20	12.40	8.30	62.45
30	11.18	9.82	72.15
40	11.72	9.92	72.89
60	11.06	9.95	73.11

A^2—900° TO 1000° F.

Time	Wt. Loss	As_2O_3 Loss	Sulphur Loss	Burnout % As_2O_3
10	14.40	10.83	0.69	72.23
20	13.73	10.17	0.89	74.73
30	13.76	10.09	1.31	74.11
40	14.30	10.05	1.51	73.84
60	13.67	10.13	1.76	74.43

A^3—1000° TO 1100° F.

Time	Wt Loss	As_2O_3 Loss	Burnout % As_2O_3
10	11.38	0.69	79.08
20	11.07	9.81	73.26
30	11.25	9.85	72.37
40	15.33	9.92	72.89
60	13.88	9.78	71.93

A^4—1100° TO 1200° F.

Time	Wt. Loss	As_2O_3 Loss	Burnout % As_2O_3
10	13.97	9.85	72.37
20	14.06	9.89	72.67
30	14.63	10.02	73.62
40	16.00	9.73	71.42
60	16.29	9.88	72.59

A^5—1300° TO 1420° F.

Time	Wt. Loss	As_2O_3 Loss	Burnout % As_2O_3
10	16.04	7.03	58.27
20	19.15	9.10	67.52
30	23.66	10.04	73.77
40	23.32	9.01	70.61
60	22.71	8.91	65.47

A^6—1600° F.

Time	Wt Loss	As_2O_3 Loss	Burnout % As_2O_3
10	17.89	4.09	30.05
30	18.85	5.33	30.16
60	17.80	5.60	41.15

COMPOSITE % ZINC AND SULPHUR VOLATILIZED

COMPOSITE CURVE

% As_2O_3 VOLATILIZED

PER CENT. DECREASE IN WEIGHT.

It is readily seen that all roasts below 1,200° F. give practically the same percentage "burnout" of As_2O_3. Above 1,200° a lower "burnout" results, owing to combination of the arsenic with the

metallic oxides, forming arsenates. The amount of other substances burned out increases from 1,200° F. to 1,500° F., above which it decreases slightly, due to the burning in of the arsenic instead of out (composite curve). The metallic sulphates decompose, SO_2 and SO_3
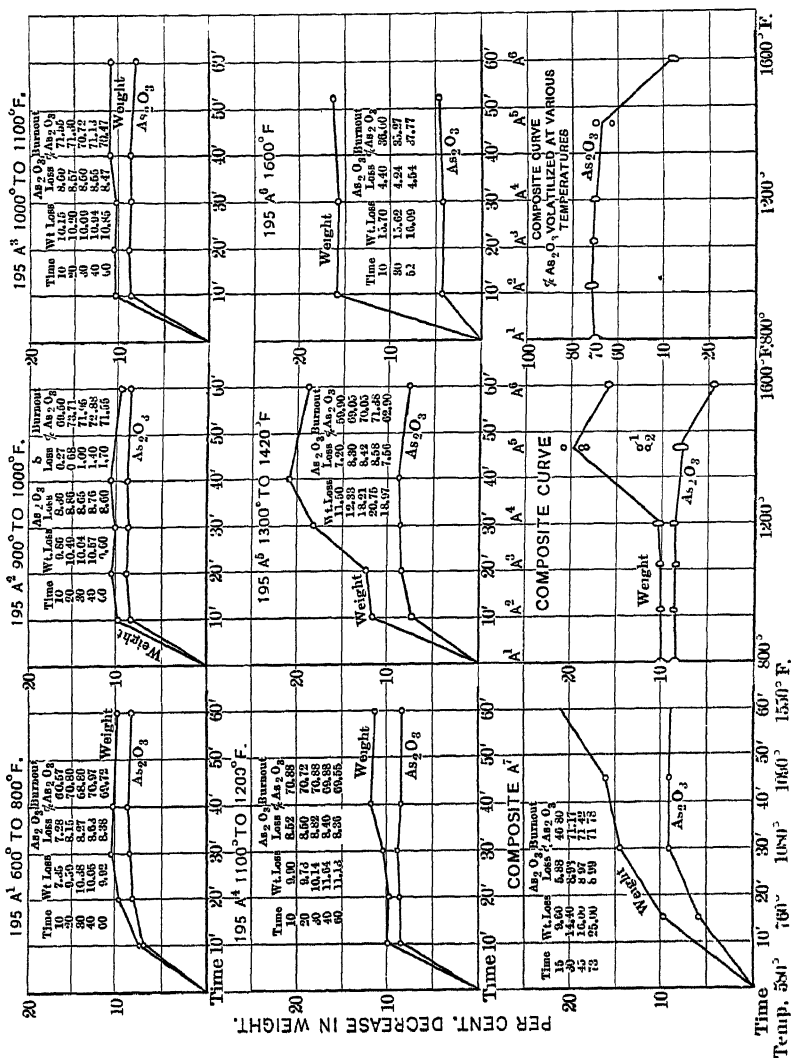
FIG. 2.—MUFFLE ROAST ON FLUE DUST. SHOWING PER CENT. OF TOTAL WEIGHT VOLATILIZED, PER CENT. OF DECREASE IN WEIGHT DUE TO As_2O_3 BURNED OUT, WITH TIME AND TEMPERATURE VARIATION, ON SAMPLE 195.

are liberated,[4] and the oxides are partly volatilized as the temperature increases.

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[4] Decomposition of Metallic Sulphates, Hofman and Wanjukow, *Trans.*, xliii., 543 (1912).

Extract from Hofman and Wanjukow's Table.

Metallic Sulphates.	Temperature of Beginning of Decomposition.		Temperature of Energetic Decomposition.	
	Degrees.		Degrees	
	C.	F.	C.	F.
FeSO$_4$,	167	337	480	896
Fe$_2$O$_3$ 2 SO$_3$,	492	917	560	1,040
PbSO$_4$,	637	1,179	705	1,301
CuSO$_4$,	653	1,207	670	1,238
ZnSO$_4$,	702	1,296	720	1,328
2 CuOSO$_3$,	702	1,296	736	1,357
3 ZnO 2 SO$_3$,	755	1,391	767	1,413

Fulton's *Principles of Metallurgy*, p. 223, gives an interesting paragraph on the roasting of arsenic and antimony ores. As it applies more or less to the roasting of flue dust in arsenic furnaces, it is here quoted in full:

"Arsenic or antimony may be present in ore to be roasted, often in the form of sulphides, or as arsenides or antimonides of other metals and sometimes in very complex mineral form. The sulphides of arsenic As$_2$S$_2$ (realgar) and As$_2$S$_3$ (orpiment) are readily volatile, and Sb$_2$S$_3$ (stibnite) is also volatile, but not to the same extent as the arsenic sulphides. Some of the arsenic and antimony is eliminated in the early stages of the roasting by the formation of these sulphides, aided by the distillation of sulphur from the pyrites present. The lower oxides of these elements form As$_2$O$_3$ which is readily volatile at 218° C., and Sb$_2$O$_3$ which is volatile only at a low red heat. Considerable of the arsenic and antimony may be eliminated in this way. Further oxidation, however, changes these oxides into the higher oxides, As$_2$O$_5$ and Sb$_2$O$_5$, which when present alone are again readily dissociated at a full red heat, but in the presence of certain base metal oxides, such as iron and copper, are converted into very stable arseniates and antimoniates of these metals, which will persist as such in the roasted material. In the process of roasting, the ore particles, as already mentioned, are alternately on the surface of the ore bed subject to an oxidizing atmosphere, and submerged among partially roasted particles from which sulphur is distilling. This sulphur vapor and SO$_2$ gas will reduce arseniates and antimoniates and again cause the volatilization of sulphides of arsenic and antimony, and their lower oxides. The arsenic and antimony are therefore most readily eliminated by alternate oxidation and reduction. A roasting furnace may be specially operated in this way, by methods of firing, when much arsenic or antimony is present."

The last part of the paragraph suggests the possible recovery of

additional As_2O_3 by roasting with pyrites. Several roasts were made with varying amounts of pyrites, to see if a better recovery could be obtained. No better recovery resulted than when roasted without pyrites. Then the residues from the high-temperature roasts were mixed with 20 per cent. of pyrites and 100-g. samples of the mixture were roasted at various temperatures up to 1,600° F. The samples were frequently rabbled, the object being to reproduce furnace conditions as nearly as possible. There was a decrease in weight, due to the roasting of the pyrites; but the total amount of arsenic contained in the sample remained undiminished. Powdered coke was substituted for pyrites, with the same results. The conclusion reached was that arsenic in flue dust, when once "burned in" is not again broken up at temperatures below 1,600° F.; also that about 30 per cent. of the arsenic is present as compounds, stable at temperatures below 1,600° F. It must be borne in mind that the amount of available arsenic in flue dust varies. The above applies to Anaconda dust. (Certain samples of Midvale dust gave 90 per cent. of available As_2O_3.) The best temperature indicated for furnace operation is 950° F. At this temperature the As_2O_5 and As_2S_3 break down and As_2O_3 is volatilized. Only a small amount of SO_3 is liberated, and the metallic oxides have a very low vapor pressure at this temperature.

Gas Saturation.

While at the University of California, H. V. Welch determined the vapor pressure of arsenic trioxide up to 400° F. (His results are to be published shortly.) Using the same character of apparatus, the vapor pressure of As_2O_3 from flue dust was determined at 833° F., the boiling point of sulphur. We are not absolutely certain about the exact location of this last point, due to the use of flue dust instead of pure As_2O_3. However, the range of error attained is most probably within the range of accuracy of the curve plotted. Using these two points, a logarithmetic vapor-pressure curve, Fig. 3, was constructed.

The vapor-pressure curve is plotted as a gas-saturation curve, and shows the amount of arsenic trioxide a cubic foot of gas will contain when heated in the presence of sufficient As_2O_3 to saturate it at atmospheric pressure. It will always be impossible to get complete saturation in actual practice in the furnace. However, when the hot gas leaves the arsenic furnace, only partly saturated, it will not give up its load of arsenic until the temperature decreases to the saturation point; upon further cooling, the As_2O_3 is condensed, the amount

remaining in the vapor state being that indicated by the curve for that temperature. When cooled to 250° F. only a trace remains in the vapor state. This may be taken as the temperature necessary for complete condensation.

This suggests that the nearer the gas approaches to saturation upon leaving the furnace, the smaller will be the amount of gas to be heated, and later cooled to condense the As_2O_3. In other words, the gas volume is proportional to the fuel bill, after allowing for radiation, heat carried out in the residue, and latent heat of vaporization of the As_2O_3.

Anaconda Arsenic Plant.

The first refining is done in wood-fired, revolving-hearth reverberatories of the Brunton type. The arsenic plant feed is the dust from the last 100 hoppers under the 120-ft. flue, which in reality is a large dust chamber, in which the velocity is decreased to 11 ft. per sec.
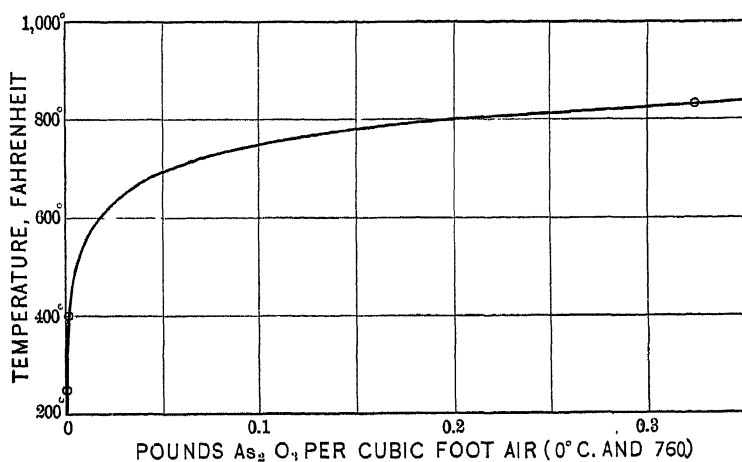
FIG. 3.—ARSENIC TRIOXIDE SATURATION CURVE.

The Brunton furnace is circular, 14 ft. 6 in. in diameter, with two outside fire-boxes. The hearth revolves upon a central shaft. The dust is fed through a dust-sealed hopper on to the center of the hearth, and is slowly moved to the outer edge by three rows of stationary rabbles fastened in the roof of the furnace. There is 3 ft. of clear space between the hearth and the roof. The "residue" is dropped into receiving hoppers and furnishes a regular part of the copper reverberatory feed.

The hot flames from the fire boxes pass directly over the charge, heating it and carrying out the As_2O_3 vapor through brick flues to the condensing chamber. The condensing chamber is 8 ft. high, 16 ft.

wide, and 240 ft. long. It is connected with the main flue system to provide draft and to receive any flue loss. The main condensing chamber is divided into small rooms, or kitchens, by cross walls at intervals of 7 ft. These kitchens are connected by openings through the cross walls, so arranged as to give the gas a zigzag course, causing it to come in contact with the cold walls and ceiling, on which the As_2O_3 crystals form. There is an opening from the outside into each kitchen for the removal of the crude arsenic. These openings are closed with wooden doors while first refining is going on. At intervals of three or four months the Bruntons are closed down and the kitchens opened; the crude arsenic is shoveled into wheelbarrows and taken to the refining furnace.

Refining.

The " refining " furnace is a small coke-fired reverberatory; hearth area, 10 by 20 ft. It has three hoppers, from which the " crude " is dropped into the furnace in 1-ton charges. There is one fire box, in which coke is burned. The hot gas from the burning coke passes over the charge, volatilizing the As_2O_3 and carrying it to the refining kitchens. These are similar in all respects to the first-refining kitchens.

When through refining, the kitchens are opened and the product taken in wheelbarrows to the pulverizer, where it is ground and barreled for market. The grinding is done in a closed room with glass windows, through which the operation can be watched. The dust enters the barrels through a leather spout. Many precautions are taken to prevent dust and protect the laborer from arsenic poisoning.

Midvale Arsenic Plant.

The Midvale arsenic plant was originally modeled after the Anaconda arsenic plant. The furnaces are of the same type. The kitchens are of the same pattern. Iron doors are used instead of wooden ones, to give additional radiation. A double row of first-refining kitchens allows continuous operation. The feed for the arsenic plant is the dust from the bays of the blast furnace bag house. It averages about 30 per cent. of arsenic and often goes as high as 50 per cent. The dust when fed to the Bruntons had a tendency to fuse and ball up on the hearth and rabbles. This formerly caused a lot of trouble. By controlling the temperature and feeding pyrite with the dust, the tendency to fuse was overcome, and a larger capacity and better recovery is claimed. The pyrite furnishes a large source of heat at the point where it is most needed. It keeps the charge open and prevents flowing and sintering on the hearth. The sulphur vapor and

SO_2 may reduce a portion of the arsenic present as arsenate; although this is not indicated by our laboratory experiments.

The first kitchen of each line is equipped with a Bristol recording pyrometer. The temperature is never allowed to go above 650° F. This means about 950° in the furnace (the ideal temperature for maximum recovery). A bag system prevents flue loss. Draft is supplied by an electrically driven 70-in. Sturtevant blower. The draft is equivalent to $\frac{5}{16}$ in. of water, measured at the fan. The pressure on the bags is equivalent to 2 in. of water, also measured at the fan.

The second-refining furnace, like the Anaconda furnace, originally had three charging hoppers. The third hopper from the fire box was removed and the opening closed. This shortened the time necessary to volatilize a charge of " crude." (The third hopper charged crude into the cold end of the furnace.) Powdered lime, about 1 in. thick, is spread on the hearth before starting refining. This prevents the crude from sticking to the bottoms and facilitates the removal of sinter. The line of kitchens was too short to effect the necessary cooling, so cold air was let in through openings covered with burlap. The burlap prevents outside dust from being drawn into the kitchens. The admission of cold air caused the formation of fume, which would not settle in the kitchens. Bags were installed to prevent loss. The bags are shaken every 2 or 3 hr. When the kitchens are filled with refined arsenic the furnace is shut down. The product is shoveled into buggies and transported by overhead tram to the pulverizing room. Pulverizing and barreling are done the same as at Anaconda.

Special one-piece outer garments are provided and kept clean by the company, A small bonus is paid for work in the arsenic plant.

Experiments were undertaken to determine the per cent. of arsenic saturation of the gas and to furnish the data for a heat balance.

Midvale Heat Balance on First Refining.

	Pounds.	Temperature C. In.	Temperature C. Out.	Latent Heat.	Sp. Heat.	Pound-Calories	Pound-Calories Absorbed.	Pound-Calories Evolved.	Per Cent. Heat Absorbed.	Per Cent. Heat Evolved.
Dust	2,000									
Pyrite	338.4									
Total As₂O₃	392.9									
Coke	195.6					6,889		1,847,488		87.3
Sulphur	90.2					2.64		195,193		12.7
Residue	1,911.2	20	434		0.22		178,557		11.0	
As₂O₃	84.9			70			5,943		0.4	
Gas	33,546.2	20	149		0.2375		1,027,772		66.6	
Radiation a							380,409		22.0	

a By difference.

This calculation is based on the amounts of coke and sulphur burned per ton of dust treated as shown by the 1912 arsenic plant balance. The volume of gas was measured at the fan, where the temperature was 149° C. The As_2O_3 indicated in the table is the amount that would be in the vapor state in the gas volume at 149° C. and would still retain its latent heat of vaporization. The weight of residue is the residue from treating 2,000 lb. of dust and 338.4 lb. of pyrites. The sulphur represents that burned from the pyrites, and is assumed to be burned to SO_2. As the "sulphur burned" is about one-half of the total sulphur present, it is assumed to represent the loosely bonded atom of the FeS_2, and no allowance is made for the heat of decomposition.

The measured temperature in the furnace was 510° C. Since the volume of gas cooled from 510° to 148° C. and the radiation was only 22 per cent., it is obvious that there was a large amount of cold air mixed with gas to help cool it. 195.6 lb. of coke burned to CO_2 will only heat 10,996 lb. of air to 510°, allowing 5 per cent. for furnace radiation. (The heat from the burning sulphur counterbalances the heat carried out in the residue.) The 10,996 lb. of gas was then used to carry out the 392.9 lb. of arsenic trioxide. The gas at 510° C. was less than 1 per cent. saturated (see gas-saturation curve).

The above calculation points to a furnace giving the best possible contact between the hot gas and the flue dust, to give better saturation.

Conclusion.

The experiments, calculations and observations suggest certain points, which if embodied in arsenic plant construction and operation should greatly improve recovery and reduce costs.

The temperature in first refining should be about 950°, and under no consideration should be allowed to exceed 1,200° F.

The gas must be cooled to 250° F. before the As_2O_3 is completely condensed.

Furnaces.

Nearly any furnace will give a good recovery if properly operated; but to reduce costs as well, the furnace should have as many of the following points as possible : A large capacity; easy heat and draft control; good transference of heat to the charge; large hearth area, with means for constantly shifting surface of the dust; intimate contact of the hot gas with the dust, preferably a constant falling of the hot dust through a slow-moving current of the hot gas; continuous operation, with sealed feed and residue-discharge openings; gradually increasing temperature of the dust towards the residue-discharge end; a decreas-

ing gas temperature to the saturation point at the gas-discharge end; simple operation; and low cost of upkeep.

The length of time the dust should remain in the furnace depends upon the thickness of the bed of dust and the rabbling it receives. If given an opportunity to volatilize, the As_2O_3 will be eliminated in from 30 to 45 min.

Rotating cylinder furnaces of the Oxland or White Howell type seem to fulfill most of the above points. The tendency to dust could be overcome by employing the scheme suggested by Rothwell;[5] i. e., to construct a dust chamber at the end of the furnace, so that the dust would slide back into the furnace.

Any of the multiple-hearth furnaces could be remodeled to make good first-refining furnaces. Cast-iron hearths could be used to give better transference of heat. At 950° F. the rabble arms and castings will stand up without cooling.

Cooling or Precipitation Chambers.

There are two general methods, either of which has advantages for the separation of As_2O_3 from the furnace gas.

1. *Cooling Followed by Bags.*[6]—The gas from the furnace is mixed in a cooling chamber with cold air, to lower its temperature to 250° F. It is then filtered through bags to separate the resulting fume. The product in this case is a fine dust, and can be drawn from hoppers. No expensive flue system is required; the filtered gas can be discharged in the building.

Constant attention is required to look after the motor, shake the bags and regulate the cold-air supply. Coke must be used in the furnaces, as soot and moisture from burning wood or coal spoil the bags and contaminate the arsenic.

2. *Precipitation by Decreasing the Saturation Point of the Gas by Radiation.*—When using the precipitation method, the gas volume should be as small as possible. The hot gas is conducted into kitchens of the Anaconda type; the walls should be of good conducting material, and gas-tight, to prevent infiltering air. The number of kitchens must be sufficient to give proper cooling in summer time, to prevent excessive loss. Where this method is used, there is always a small flue loss due to the formation of fume. The gas, upon coming in contact with the cold surface, is chilled, the saturation point of the gas is decreased, and the As_2O_3 precipitates in crystalline form on the cold surfaces.

[5] *Mineral Industry*, vol. i., p. 234 (1892).
[6] Midvale practice.

An extra set of kitchens is necessary to permit continuous operation, because the chambers must be entered to remove the deposit from the walls.

Second Refining.

The reverberatory refining furnace as operated at Midvale gives excellent results. Better transference of heat could be effected by making the hearth of cast iron and having the hot gas enter and travel the length of the furnace underneath the hearth, returning over the top of the charge to carry out the As_2O_3 vapor.

Notes on the Great Falls Electrolytic Plant.

BY WILLIS T. BURNS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

CONTENTS.

I. INTRODUCTION.

These notes are submitted, not as a discussion of the modern practice of electrolytic-copper refining, but as the record of a refinery that was among the pioneers in the field and that is to-day, and has been for 17 years, operating under conditions not to be found in any similar plant in the country. I refer to the current density, which is at least 60 per cent. higher than that regularly employed elsewhere, and to the fact that the anodes treated are the direct product of the converter rather than of the reverberatory furnace. The writer has no knowledge of any other refinery treating anodes of this character.

II. HISTORY.

This refinery, now one of the oldest plants of its kind operating in this country, was built in 1892 by the Boston & Montana Consolidated Copper & Silver Mining Co. to refine the product of its Great Falls reduction works. The estimated capacity of the plant as designed was 65,000 lb. of cathodes per day at a current density of 16 amperes per square foot.

The plant produced its first cathodes in February, 1893, and has been in almost continuous operation since that date. The output of the plant for the year 1893 was 3,179 tons. As the smelter output increased demands for a larger refinery output were made and as no appropriation for enlarging the plant was forthcoming increased production could be obtained only by increasing the current density. As an abundance of water power was available and preliminary experiments had demonstrated that cathodes, of satisfactory quality, could be economically produced at more than double the density the plant was then operating at, new and larger generators were installed in 1896 and the current density brought up to 40 amperes per square foot. Since that date the current density has been reduced by adding 76 refining tanks and by increasing the number of electrodes per tank.

In 1893, 3,179 tons of copper were refined at an average current density of 10 amperes per square foot; the production for the year 1912 was 31,596 tons and the average current density 34.1 amperes per square foot.

III. GENERAL DESCRIPTION OF PLANT.

The refinery was built at the reduction works of the Boston & Montana Consolidated Copper & Silver Mining Co. situated at the Black Eagle falls of the Missouri river three miles from the city of Great Falls, Mont. Power for the operation of the reduction works and the electrolytic plant was obtained from water power developed at this point.

As originally designed and constructed the refinery consisted of the tank house, a brick building 171 ft. 6 in. by 108 ft. (Fig. 1) containing 288 electrolytic-refining tanks of the following inside dimensions: 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. deep, built of Western pine. The sides and bottoms were of 3-in. lumber and the ends of 2-in. material. The tanks were tied at the ends with 3 by ½ in. iron straps held together by 1-in. rods as shown in Fig. 2. The tanks were lined with ⅛-in. chemical sheet lead weighing 8 lb. per square foot.

The tanks, which were supported in place by two 8 by 8 in. timbers which rested on stone piers 18 in. high, were insulated from the timbers by 4 by 4 by 1 in. glass insulators, eight under each tank. Had 5 or 6 ft. of head room, instead of 18 in., been provided under the tanks, subsequent operations would have been greatly facilitated.

The tanks were arranged in 18 double rows, eight tanks deep (Fig. 1) with an aisle between each double row. The tanks as originally built were all on the same level, making it necessary to establish an individual system of circulation of the electrolyte for each tank. This was accomplished by providing two circulating tanks for each section of 96 tanks;

PLAN OF TANK BUILDING

SECTION C-C

DETAIL OF TANKS

PLAN OF BOILER HOUSE

SECTION A-A

SECTION B-B
SULPHATE BUILDING

SECTION OF TANK BUILDING ON CENTER LINE
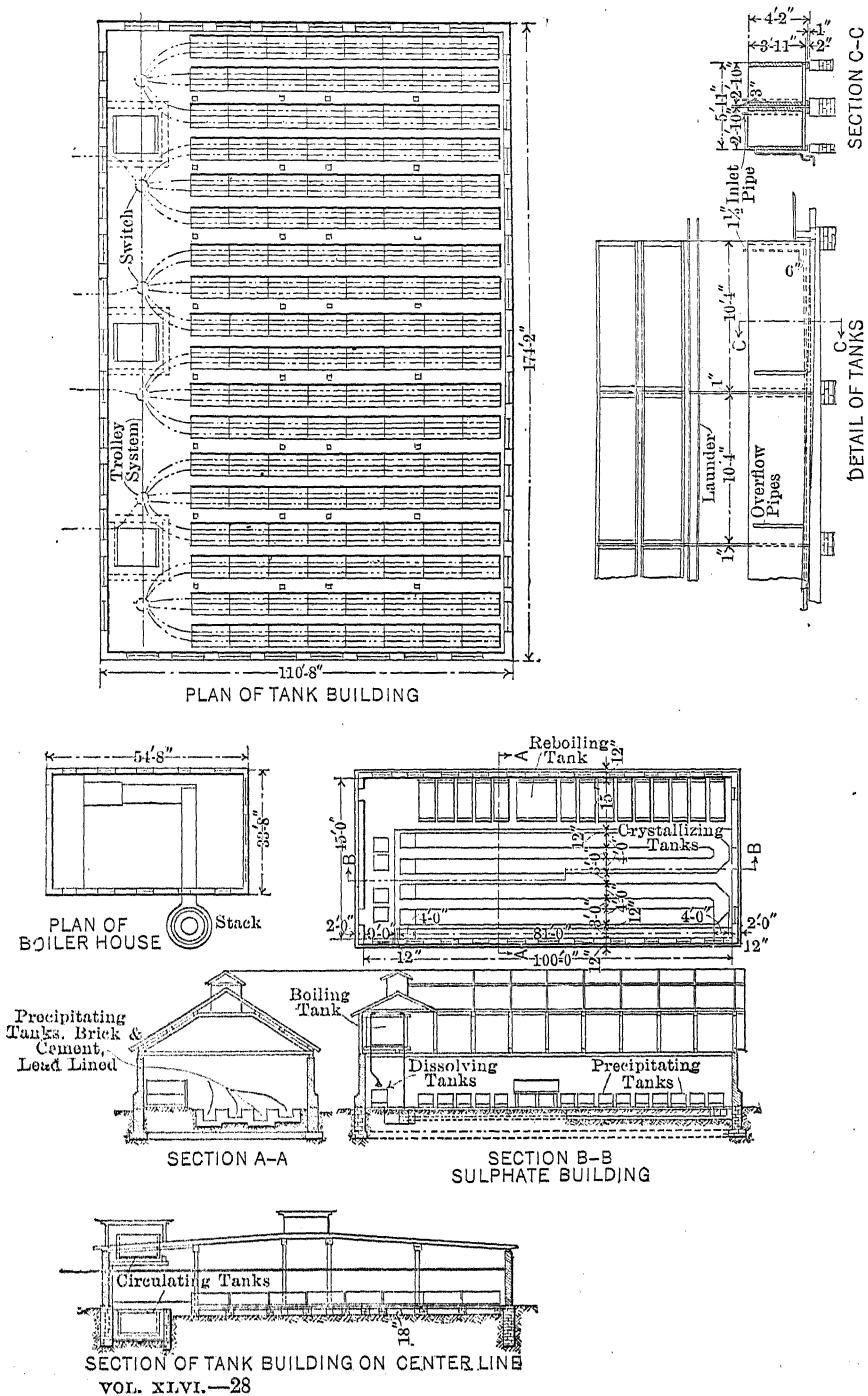
FIG. 1.—GREAT FALLS ELECTROLYTIC PLANT AS ORIGINALLY BUILT.

these tanks were 10 ft. square by 8 ft. deep, lined with 12-lb. chemical sheet lead. One of these tanks was placed above the level of the refining tanks and one below, as shown in sketch. From the overhead tank the electrolyte was delivered, through an overhead system of lead pipes, to each separate refining tank. The solution was discharged into the refining tank at one end, near the bottom of the tank, and was allowed to overflow through a 1-in. lead pipe entering the tank 3 in. from the top near the other end. This pipe discharged into an open launder leading to the lower circulating tank, from where it was raised to the overhead tanks by various devices to be described later. This system, which permitted only a periodical circulation of the solution, soon gave way to a more efficient system to be described further on. Heating the electrolyte was effected by passing live steam through 125 ft. of 1-in. 8-lb. lead pipe in each of the overhead circulating tanks.

A ¾ by 4 in. steel trolley track was suspended from the roof timbers over each row of eight tanks. The anodes and cathodes were removed in and out of the tanks by means of chain blocks suspended from two-wheel trolleys on this track.

The current was distributed by means of ¾ by 4 in. rolled-copper busbars attached to the sides of the tanks with wooden brackets. The tanks were all connected in series and the electrodes in multiple.

· The original installation included a sulphate plant for the manufacture of bluestone and a boiler house containing one 100-h.p. locomotive-type boiler to furnish steam for heating and boiling solutions.

The sulphate plant, shown in Fig. 1, was well appointed for the manufacture of bluestone as described later. The capacity of this plant was 2,500,000 lb. of bluestone per year.

The above is a brief description of the plant as it originally existed. Fig. 3 is a sketch of the plant as it exists to-day. One division of 32 refining tanks has been added to the main tank room, 44 new electrolytic-refining tanks for the production of starting sheets have been housed in additions as have the 10 new insoluble-anode tanks shown in sketch. An addition has been built south of the tank room in which is installed the equipment for pumping and heating the electrolyte. An addition has been built on the east containing the scale house and a 7-ton traveling crane for handling the cathodes and scrap after weighing. The addition on the north side of the building contains store room, shops and office. Fig. 4 is a flow sheet of the electrolytic plant.

In the new building south of the bluestone building are the carpenter and lead-burning shops and the machinery used in trimming sheets. The four large round tanks situated further south are for the storage of solutions.

The original boiler house has been replaced by a brick building con-
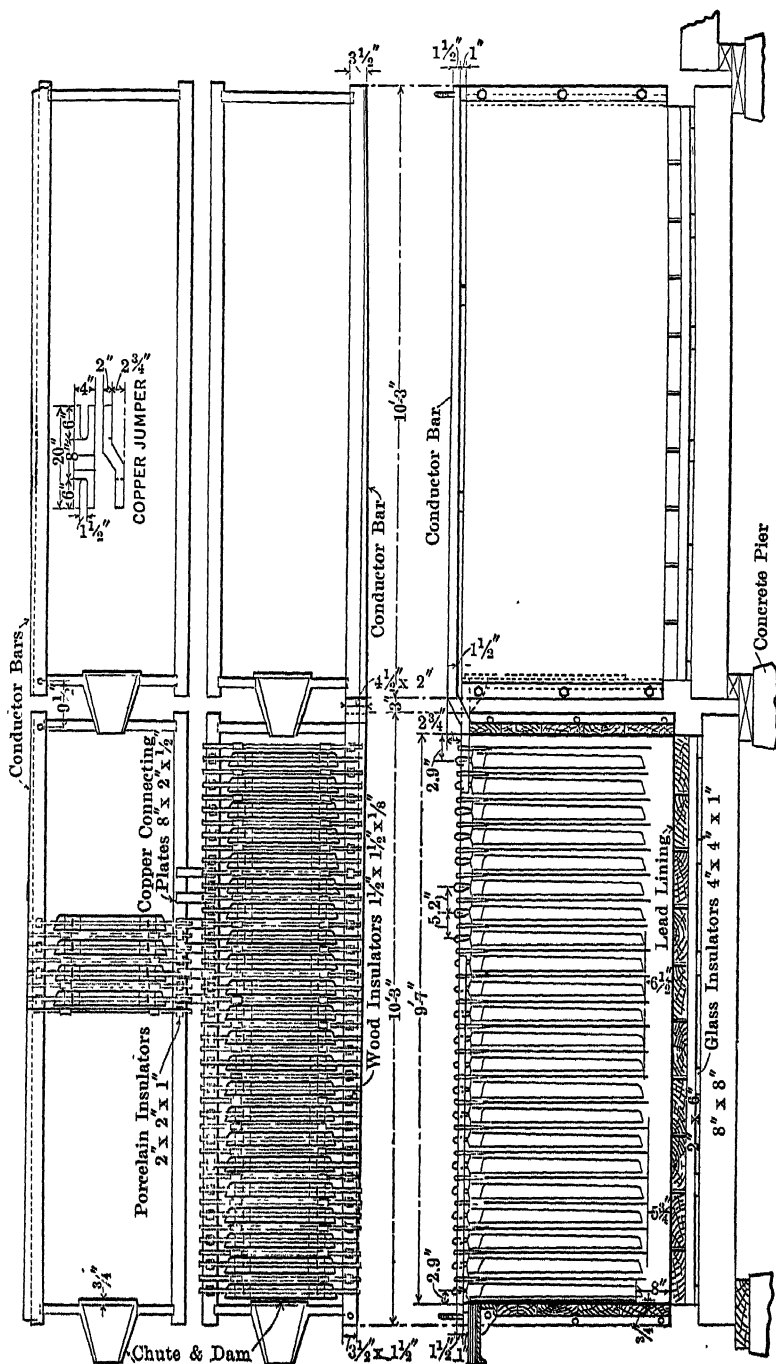
FIG. 2.—TANK DETAILS.

taining two 150-h.p. Heine and one 100-h.p. Scotch marine boiler from which steam for heating and boiling solutions is obtained.

As previously stated the refining tanks, as originally installed, were all on the same level and to obtain a circulation of the electrolyte it was necessary to deliver the electrolyte to each tank through a separate pipe line. This system was found unsatisfactory and the refining tanks were rearranged in a cascade of eight tanks with a drop of 2¾ in. between
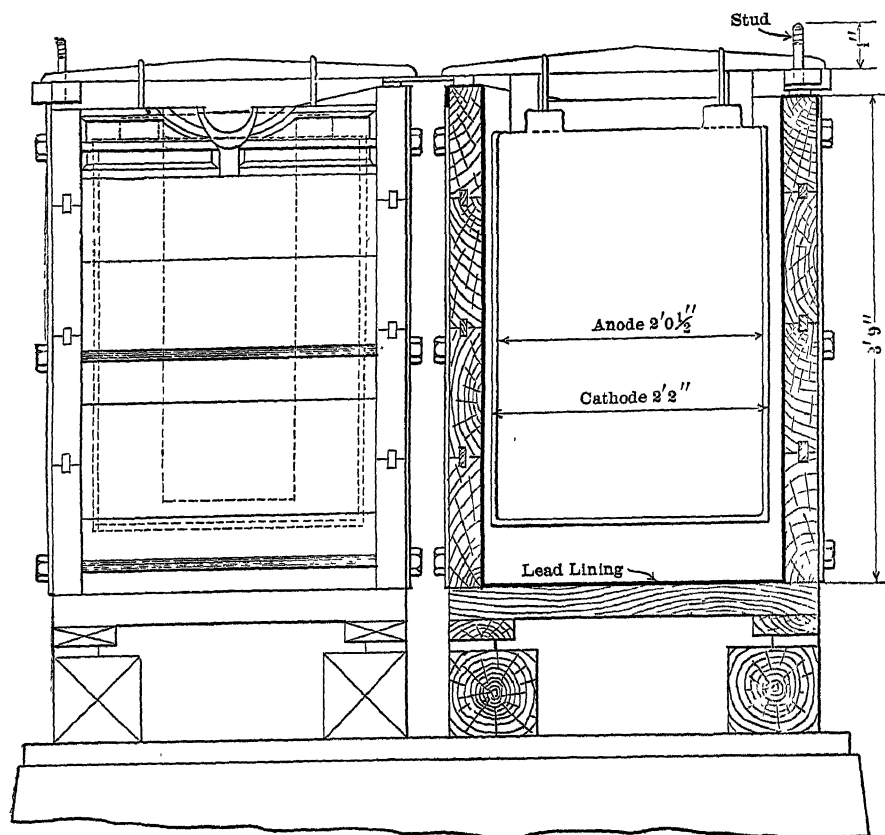
FIG. 2A.—TANK DETAILS.

tanks as shown in Fig. 3. The electrolyte is now delivered to the upper tank of the cascade at the surface and is drawn off from a point 8 in. from the bottom at the lower end of the tank by means of the dam shown and is delivered to the next tank through the cast-lead and antimony chute shown in Fig. 2. From the last or lower tank of the cascade the electrolyte is returned to the pump room through a 2-in. 7-lb. lead pipe attached to the sides of the tanks.

PLAN OF TANK BUILDING

SECTION B-B   SECTION C-C

SULPHATE BUILDING

SECTION OF TANK ROOM ON LINE A-A
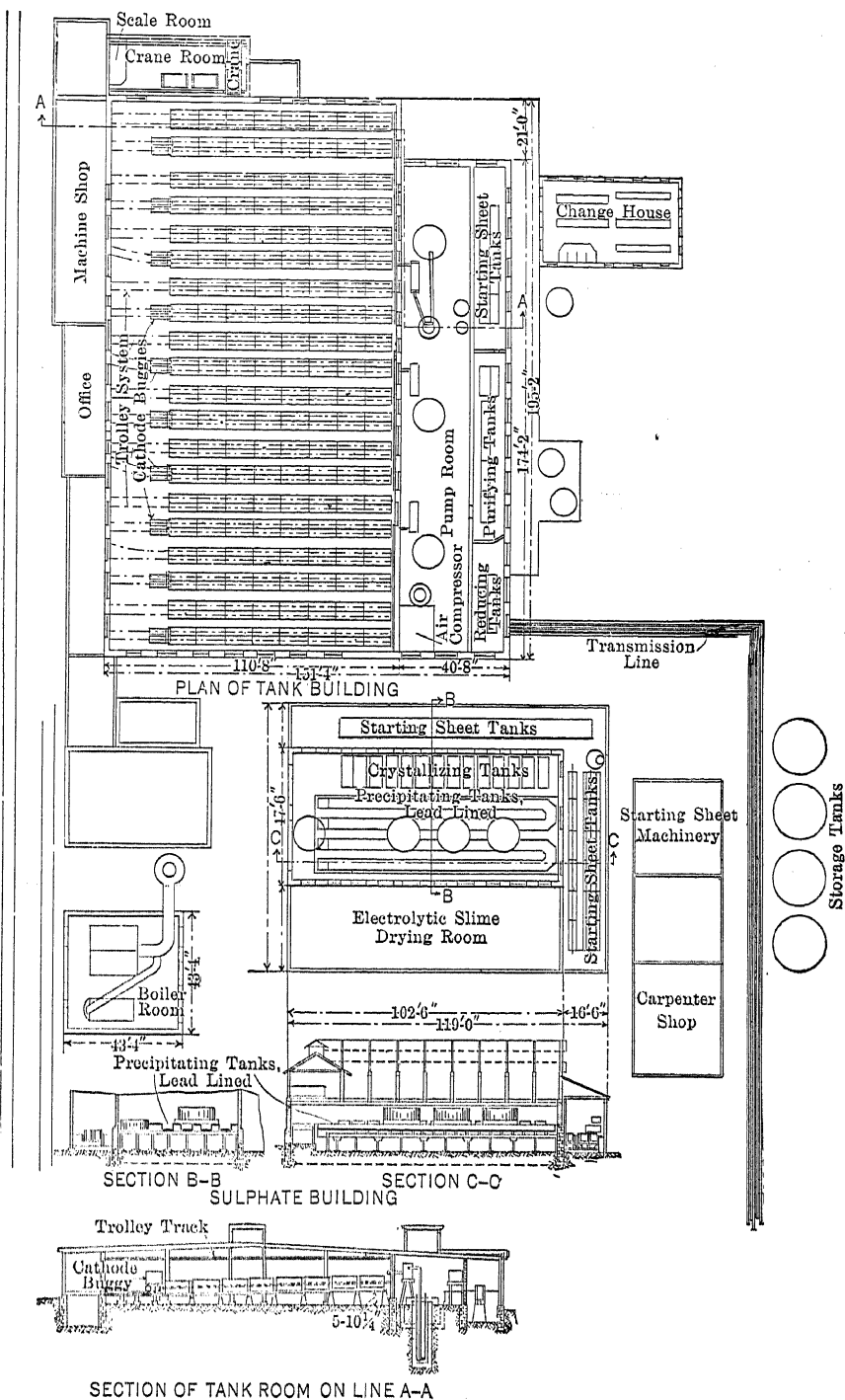
FIG. 3.—GREAT FALLS ELECTROLYTIC PLANT AS REMODELED.

The original refining tanks were lined with $\frac{1}{8}$-in., 8-lb. chemical sheet lead and the overflow consisted of a 1-in. lead pipe burned to the lining. Tanks constructed in this manner were found to last about 10 years in service. As they gave out they were replaced by tanks of the same dimensions lined with $\frac{1}{8}$-in., 7-lb. lead containing 6 per cent. antimony. The cast chutes of the same material were built into the ends of the tanks. Tanks thus constructed have now been in service for 11 years and do not show much deterioration. The antimonial lead is harder and
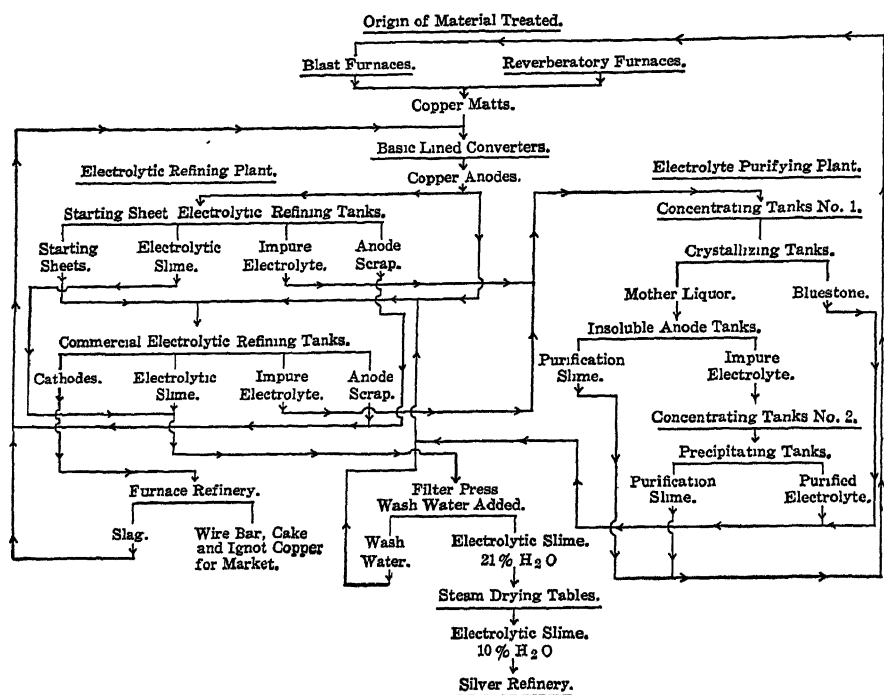
FIG. 4.—FLOW SHEET OF GREAT FALLS ELECTROLYTIC PLANT.

has less tendency to creep and buckle under extreme changes of temperature than is the case with the chemical lead. Practically all of our tanks are now lined with antimonial lead.

When the change from 4,000 to 9,000 amperes was made in 1896 it became necessary to increase the size of the tank busbars. The $\frac{3}{4}$ by 4 in. rolled-copper conductors were removed from the sides of the tanks and a taper bar $4\frac{1}{2}$ by 2 in. in the center and $1\frac{1}{2}$ by 3 in. at the ends was laid on the top of the tanks, as shown in Fig. 2. The bars were cast at our works from wire-bar copper. These bars are insulated from the tanks by means of $\frac{1}{2}$ by 4 in. pine strips previously boiled in pine tar. These strips are renewed once in six months.

Into the ends of the busbars ⅞ by 6 in. iron studs are screwed to receive the copper shunt used to cut four tanks out of circuit for removal of slime.

The anode and cathode rods and the connection plates, used between the tanks, are also cast at our furnace refinery in copper molds. Details of tank connections are shown in Fig. 2 and Fig. 2A.

IV. ANODES.

As previously mentioned this refinery is the only plant of its kind treating converter-copper anodes. The general practice elsewhere is to
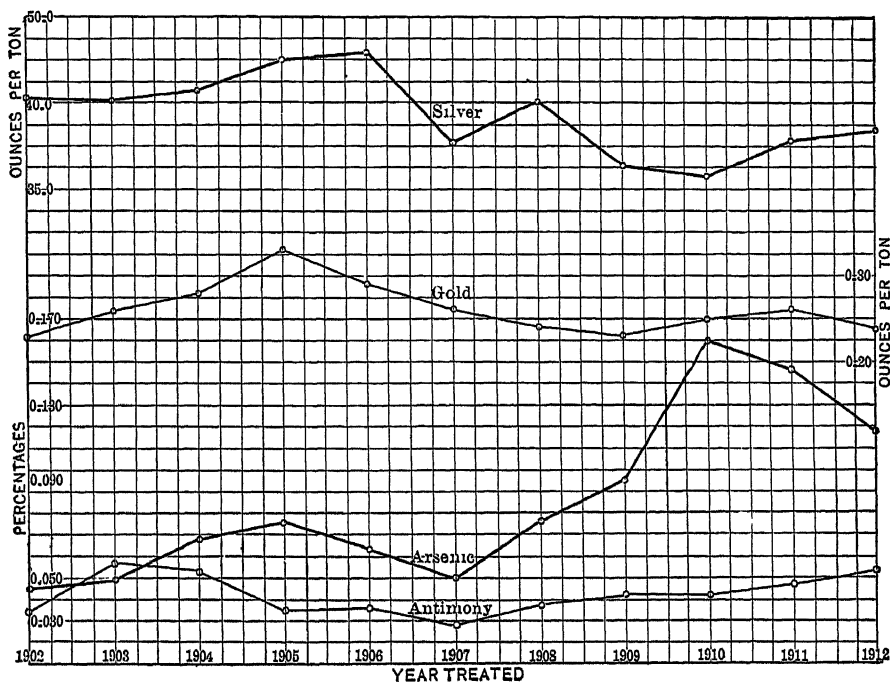
FIG. 5.—CONVERTER ANODES.

transfer the molten copper from the converter to the reverberatory furnace. Here the necessary oxidizing and poling of the copper to bring it to the proper pitch for casting takes place. The copper is then cast into anodes by means of a suitable casting machine.

This refining process has the effect of increasing the copper contents of the metal by from 0.3 to 0.4 per cent., sulphur dioxide being the principal impurity eliminated. The resulting anode is much denser than the converter anode and the casting is free from the uneven surfaces which appear on the converter anode. The analyses of anodes are shown in Table I and Fig. 5.

TABLE I.—*Analyses of Anodes, Electrolyte, Wire Bar and Electrolytic Slime.*

	Converter Anodes Per Cent.	Electrolyte Per Cent.	Wire Bar Per Cent.	Electrolytic Slime Per Cent.
Copper.........	99 1300	3.280	99 9500	43.3400
Arsenic.....	0.1183	0.500	0 0016	3 0300
Antimony	0 0534	0.041	0.0015	3 4600
Nickel........	0.0420	0.377	0.0006	0.0800
Cobalt..	0.0018	0 016	Trace	0 0060
Bismuth.....	0.0038	0.021	0 0004	0 1100
Iron........	0 0110	0.600	0.0006	0 3640
Silver	0.1371	None	0.0030	17.1870
Gold........	0.0008	None	Trace	0.1200
Selenium.....	0.0090	None	....	1 2000
Tellurium.	0.0170	None	....	2 1000
Lead..........	0 0065	Trace	Trace	0 7600
Zinc..............	0 0035	0.418	0.0001	0.0900
Sulphur.............	0.2610	....	0 0025	13.2100
Oxygen.........	. . ..	....	0.0350	....
Silicon........	... .	. . ..	... .	0 1770
Chlorine......	. .	0.0040	...	0 0260
Carbon......	. .	. ...	....	0 5900
Platinum	. .	. ...	.. .	0 000166
Free sulphuric acid	. .	13 0300	.. . .. .	....
Specific gravity..	. ...	1 220	. ... . .	. . ..

Table II shows one of a number of comparisons, made at this plant, between converter and refined anodes. In conducting this test three divisions, of 32 refining tanks each, were employed. One-half of the tanks in each division were treating refined anodes while the other half contained converter anodes. By this arrangement the effect of the personal equation of the men in charge of the tanks was reduced to a minimum.

It will be noted that while the ampere efficiency of deposit was 3.6 per cent. higher in the refined-anode tanks than in the tanks containing converter anodes the production per kilowatt-hour was slightly higher in the converter-anode tanks. Subsequent experiments have shown that the refined anodes admit of closer spacing than do converter anodes, without the same reduction in ampere efficiency, the effect of which is to decrease the resistance and increase the production per kilowatt. The difference, however, was not found to be great.

The advantages in favor of the refined anodes, as they appear in Table II, are: First, higher grade of slime resulting in a saving in cost of slime refining; second, lower silver contents of cathodes, all of the silver in the cathode representing a dead loss; third, the lower percentage of anode scrap.

When reduced to dollars per ton of copper refined the sum of the

savings that would result from the use of refined anodes, in a plant of this type, was found to be less than one-half the cost per ton of the reverberatory-furnace treatment of the copper.   The cathodes produced from the two types of anodes differed but little in physical appearance.

TABLE II.—*Comparison of Converter and Refined Anodes Cast in the Same Molds.*

	Converter Anodes	Refined Anodes
Number of days covered by test	50	50
Number of refining tanks employed . .	48	48
Average Analyses of Anodes:		
Per cent. Cu.	98 91	99 27
Per cent. As+Sb.........	0 072	0 071
Oz. Ag per ton................	59 09	61 14
Oz. Au per ton.	0.200	0.219
Average Analyses of Electrolyte:		
Specific gravity................	1 20	1 20
Grams per liter Cu......	43.5	43 5
Grams per liter free acid.	160	160
Grams per liter As............	11.97	11.97
Grams per liter Sb...................	0.49	0.49
Grams per liter Fe....	10 09	10 09
Grams per liter Cl.........	0.045	0 045
Average Temperature of Electrolyte:		
Inlet of 8-tank cascade, C°.....	58	58
Outlet of 8-tank cascade, C°.	54	54
Rate of circulation of electrolyte, gal. per min.. . ..	6	6
Number of anodes per tank.........	20	20
Number of cathodes per tank...	20	20
Average weight per new anode, lb.....	525	632
Average thickness per new anode, in...	3	3
Distance, center of anode to center of cathode, in..	2.87	2.87
Active cathode surface per tank, sq. ft............	252	252
Average amperes per tank..........	8,387	8,387
Average volts for 48 tanks............	27 21	28.53
Average volts per tank..........	0.567	0.594
Average kilowatts for 48 tanks....	228 2	239.3
Total copper deposited, lb.....	1,103,749	1,148,749
Average age of cathodes drawn, days............ ..	2½	2½
Average ampere efficiency of deposit, per cent......	88 3	91.9
Average amperes per sq. ft. cathode surface.......	33.3	33.3
Average lb. copper deposited per kilowatt-hour.....	4.03	4.00
Average oz. per ton silver in cathodes.....	1 25	.95
Average per cent. As+Sb in cathodes............	0 0043	0.0043
Average per cent. anode scrap........	8 00	5.30
Analyses of silver slime:		
Per cent. Cu.............................	40.3	18.80
Oz. Ag per ton......................	6,755.00	14,079
Oz. Au per ton.............	18.34	38.45

In a crane-operated electrolytic refinery, where the anodes and cathodes are handled in tank units and not individually as here, difficulties would result if converter anodes were used because of the fact that converter anodes corrode less uniformly than refined anodes.   Many of the converter

anodes would become scrap before the date set for drawing the scrap from a tank and an equal number of pieces would not be scrap at the appointed time for drawing.

The above comparison, together with other tests made at this plant, has demonstrated that refined anodes, while very desirable, would for this plant be an expensive luxury.

Fig. 6 shows the earlier form of anode used in this plant and the Morrow clip type[1] now used. The advantage of the clip type of anode is that the amount of inactive copper above the solution line is much less than is the case with the lug type. The average percentage of anode scrap for the year 1912 was 5.9; the average weight of the converter anode is 500 lb. The scrap resulting from each anode weighs about 30 lb.; this is resmelted in the converters. The clip type of anode also has the advantage, as compared with the lug type, of being suspended in such a manner as always to hang plumb in the tank. The standard converter anode measures 24.5 by 35.75 in.

The clip or hanger is made from a $\frac{1}{4}$-in. round copper rod 60 ft. long rolled in a rod mill from a bar cast at the refining furnace. This rod is cut and bent into the loop form, in an automatic machine, and placed in the anode mold. Both the anodes and cathodes are supported in the tanks by cast-copper rods as shown in sketch. The anodes are spaced 5.2 in. center to center. It will be noted in Fig. 6 that the anode is 3 in. thick at the top and 2.5 in. thick at the bottom. It is found that the anode corrodes more rapidly at the upper end where the current enters than at the lower end, that the wedge shape yields a lower percentage of scrap and that the anode retains its original shape for a longer time than is the case with an anode of uniform thickness.

V. CATHODES.

The starting sheets are produced in 44 standard refining tanks divided into two circuits connected in parallel in the main circuit and are thus operating at one-half the current density of the commercial refining tanks. Each tank contains 21 cathode blanks and 22 anodes. The blanks are made of $\frac{1}{4}$-in. rolled copper 27.5 by 39.5 in. below the solution line, as shown in Fig. 6. The anodes used in these tanks are of the same type as the standard anode but larger, being 26.5 by 38$\frac{5}{8}$ in.

Twelve-hour sheets are stripped at an average weight of 4 lb. each, measuring 26 by 36.5 in. after trimming. The men employed in stripping the plates do their work directly over the tanks using a traveling bench suspended from the trolley track as shown in Fig. 7. This bench, on which the sheets are piled after stripping, is moved from tank to tank as the

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[1] U. S. patents No. 621,121 and 631,471.

Original Form of Anode   Original Form of Cathode

Starting Sheet Anode   Rolled Copper
Morrow Clip Type   Starting Sheet Blank

Standard Anode   Present Form of Cathode
Morrow Clip Type   Morrow Clip Type
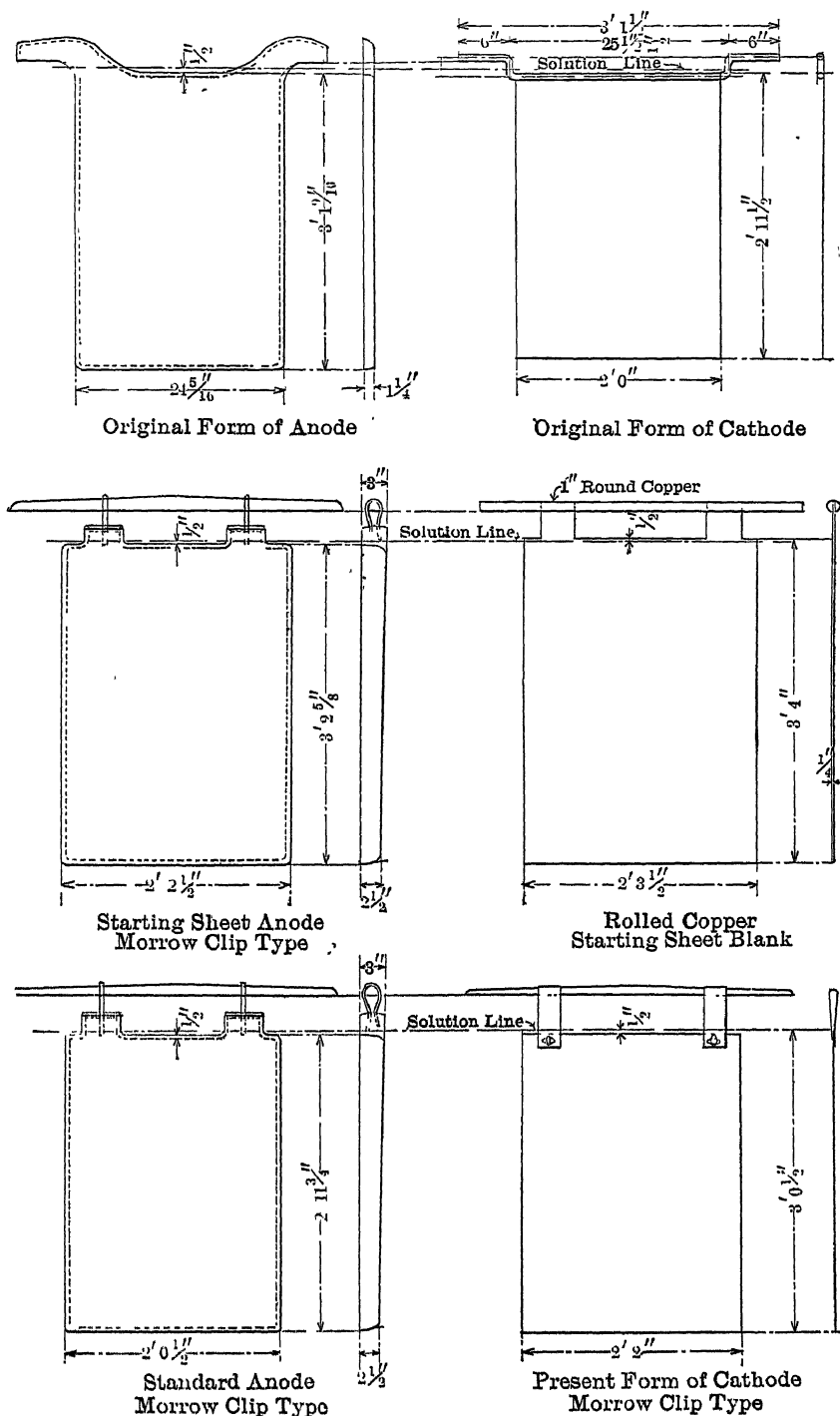
FIG. 6.—ANODES, CATHODES, AND STARTING SHEET BLANK.

men advance. The blanks are covered with a thin coating of low-grade mineral oil to which 0.5 lb. of artificial graphite per gallon of oil is added. Grooved wooden strips are used on the edges of the plates to prevent deposition of copper. The strippers work independently and each man strips and delivers to the trimmer 360 sheets per 8-hr. shift. The electrolyte used in these tanks differs slightly from that of the main tank room. An average analysis is given in Table III.
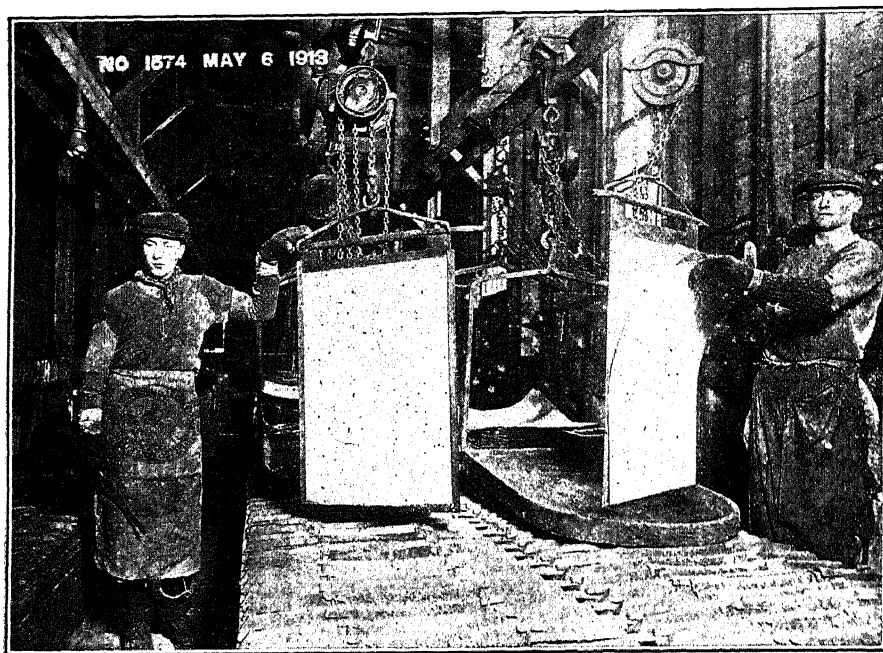
FIG. 7.—STRIPPING STARTING SHEETS.

TABLE III.—*Starting-Sheet Tank Electrolyte.*

Specific gravity	1.175
Free H_2SO_4, grams per liter	120.0
Cu, grams per liter	40.0
As, grams per liter	5.0
Sb, grams per liter	0.4
Fe, grams per liter	4.5
Cl, grams per liter	0.04

Table IV shows the effect of sulphuric acid in the electrolyte on the efficiency of deposition, the production per kilowatt-hour increasing as the acid increases. The sheets produced when the acid is about 120 g. per liter are found to be smoother and tougher than when acid is higher.

The starting sheets are trimmed on the four edges to prevent short circuits. Two clips or loops, cut from similar sheets, are then attached to each sheet by means of the Morrow clip machine[2] (see Fig. 6).

TABLE IV.—*Effect of Free Sulphuric Acid on Starting-Sheet Production.*

Converter anodes, 39 5 by 27.5 by 0.25 in. blanks, 3 in. anode; 2 5 in. center of anode to center of blank.

Circulating electrolyte at 4 gal. per minute.

AMPERES	Amperes per Sq. Ft.	Effi- ciency	E. M. F. per Tank Volts	Lb. Cu per Kilowatt- hour	Average Weight per Sheet	ELECTROLYTE		
						Grms. per Liter Free H_2SO_4	Grms. per Liter Cu	Tempera- ture C°
4,580	16.2	86.5	0.475	4.75	5.86	74	42 6	52.5
4,490	15.9	86.5	0.462	4.84	5 78	74	42 6	52
4,453	15 7	85.2	0.462	4.80	5.61	70	40 1	55
4,450	15.7	83 6	0.475	4.55	5.50	69	39.8	49
4,390	15.5	90.3	0.482	4.84	5.87	69	40 1	50
4,403	15 6	90.2	0 496	4.71	5.87	73	38 4	47.5
4,290	15.2	88.2	0.462	4.94	5.60	80	38.9	51
4,397	15.5	91 6	0.470	5 04	5.93	82	38.8	52
4,538	16.0	90.3	0.476	4.90	6.06	88	38 8	51
4,461	15.8	91.9	0.450	5.28	6.06	89	39.9	52
4,408	15.6	92.5	0 426	5 63	6.03	110	40 4	52
4,506	16.0	92.5	0 452	5.29	6.17	109	38.9	53
4,496	15.9	91 7	0.424	5.55	6.10	116	40.6	53
4,512	16.0	90.3	0.426	5 50	6.02	126	40.5	50
4,550	16.1	89.7	0.420	5.53	6 04	128	40.4	48
4,430	15.7	90.7	0.400	5.86	5.94	129	42.6	48
4,435	15 7	84.7	0.385	5.69	5.56	128	42.0	48
4,420	15.6	87.3	0.365	6.09	5.62	126	42.4	50
4,445	15.7	91.8	0.357	6.23	6.03	124	43 6	48
4,453	15.7	85.1	0.360	6.13	5 60	132	42.1	46
4,404	15.6	87.0	0 342	6.55	5.66	148	40.8	49
4,450	15.7	88.6	0.350	6.62	5.82	149	42.3	46

NOTE: The sheets produced with between 116 and 126 g. per liter of acid were the toughest and best.

The starting sheets are now ready for the tanks. It will be noted that the cathodes hang ¾ in. lower in the tank than the anode and are 1.5 in. wider than the anode. When using anodes and cathodes of the same size a cathode very rough on the edges resulted; increasing the size of the cathode has resulted in a cathode with comparatively smooth edges and an increase of 5 per cent. in ampere efficiency.

When the output of the generators is 2,000 kw. or more the best results are obtained while drawing 2- and 3-day cathodes. With a lower generator output older cathodes are drawn. The generator output is

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[2] U. S. Patent 600,498, Mar. 8, 1898.

governed by the head on the water wheels, which is dependent upon the stage of water in the river.

To determine the most economical age of cathodes and number of electrodes per tank at which to operate, the labor, the production per kilowatt-hour, the silver lost in cathodes and the impurities in the anodes and cathodes must be considered. Assuming the quality of the product

TABLE V.—*Effect of Varying the Age of Cathodes and the Number of Electrodes per Tank.*

AGE OF CATHODES DAYS	ELECTRODES PER TANK		Average Amperes	Average Amperes per Sq. Ft.	Ampere Efficiency Per Cent.	Cu per Kilowatt-hour Lb.	Cathode Analyses	
	Anodes[a]	Cathodes					Oz. Ag per Ton	As + Sb Per Cent.
4	20	20	9,300	36 9	88 0	3.93	1 32	0 0036
2	20	20	8,808	35 0	90 85	3.72	0 83	0 0030
3	20	20	8,877	35 2	89 00	3.75	0 83	0.0032
2	21	21	9,035	34 1	90 90	3.84	0.89	0.0032
3	21	21	9,223	34 8	89 40	3 87	1 02	0.0029
2	22	22	9,071	32.6	90.50	4.02	0.89	0.0033
3	22	22	9,167	33 0	88 80	4.07	0 95	0.0030

[a] Converter anodes. Average analysis of anodes: Cu, 99.13; As, 0.127; Sb, 0.055 per cent.; Ag, 33.97 oz. per ton; Au, 0.22 oz.

as regards impurities to remain satisfactory under the different conditions the cost per ton remains as the only consideration. Table V is a summary of various comparisons made between different ages of cathodes with varying numbers of electrodes per tank.

Table VI is the comparison, in detail, between 21 and 22 pairs of electrodes per tank while drawing 2- and 3-day cathodes.

VI. THE ELECTROLYTE.

The 320 commercial refining tanks are divided into three sections, two sections containing 96 tanks each and the third section 128 tanks.

Each section is provided with a separate electrolyte and circulating system. Copper and acid determinations are made daily on samples of each electrolyte. As, Sb, Fe and Cl determinations are made weekly.

The two groups of starting-sheet tanks each have a separate electrolyte which is sampled and assayed in the same manner as the commercial-tanks electrolyte.

The analysis of the electrolyte in the commercial-refining tanks is shown in Table I in terms of percentage in conformity with the analysis of the other materials shown in the table. The analysis of solutions is ordinarily reported in terms of grams per liter. With a current density

TABLE VI.—*Effect of Varying the Age of Cathodes and Number of Electrodes per Tank.*

Details of 21 and 22 pairs of electrodes per tank in Table V.

Standard Converter Anodes, Fig. 6.

	21 Anodes and 21 Cathodes per Tank	22 Anodes and 22 Cathodes per Tank
Number of refining tanks used	160	160
Copper deposited, 2-day cathodes, lb..... .	1,778,352	1,704,743
Copper deposited, 3-day cathodes, lb.... . .	1,742,678	1,889,401
Total copper deposited, lb	3,521,030	3,594,144
Spacing of electrodes, anode centers, in.. . .	5.45	5 20
Average amperes for 2-day cathodes	9,035	9,071
Average amperes for 3-day cathodes....	9,223	9,167
Average amperes per sq. ft., 2-day cathodes.. . . .	34 1	32.6
Average amperes per sq. ft., 3-day cathodes	34 8	33 0
Average ampere efficiency, 2-day cathodes, per cent.	90 9	90 5
Average ampere efficiency, 3-day cathodes, per cent.	89.4	88 8
Average drop per tank in volts, 2-day cathodes.....	0 612	0 581
Average drop per tank in volts, 3-day cathodes.....	0.600	0 562
Average copper deposited per kilowatt-hour, 2-day cathodes.......	3.837	4.021
Average copper deposited per kilowatt-hour, 3-day cathodes.............	3 871	4 075
Average copper deposited per kilowatt-hour, 2- and 3-day cathodes................	3.854	4.048
Analyses of Cathodes:		
2-day cathodes, As + Sb, per cent...	0.0032	0 0033
3-day cathodes, As + Sb, per cent .. .	0 0029	0 0030
2-day cathodes, Ag, oz. per ton	0.89	0.89
3-day cathodes, Ag, oz. per ton........... . .	1.02	0.95

	Same both set of Tanks
Analyses of Electrolyte:	
Specific gravity............................ .	1.219
Cu, grams per liter.........................	41.1
Free acid, grams per liter............	153
As, grams per liter.............	6 49
Sb, grams per liter..............	0.51
Fe, grams per liter................	5.11
Cl, grams per liter.................	0.044
Average temperature of electrolyte (8 tanks in cascade):	
Inlet tank, C°...............	57
Outlet tank, C°.......................	53
Speed of circulation of electrolyte, gal. per min.....	6
Analyses of Anodes (Converter):	
Cu, per cent................................	99.13
As, per cent................................	0.127
Sb, per cent................................	0.055
Ag, oz. per ton.............................	33.97
Au, oz. per ton.............................	0.22

of 34 amperes per square foot and a circulation of 6 gal. of electrolyte
per minute the best results, while treating converter anodes, are obtained
with an electrolyte carrying 40 g. per liter of copper and 160 g. per liter
of free sulphuric acid.

The circulation of the electrolyte is through a cascade of eight tanks
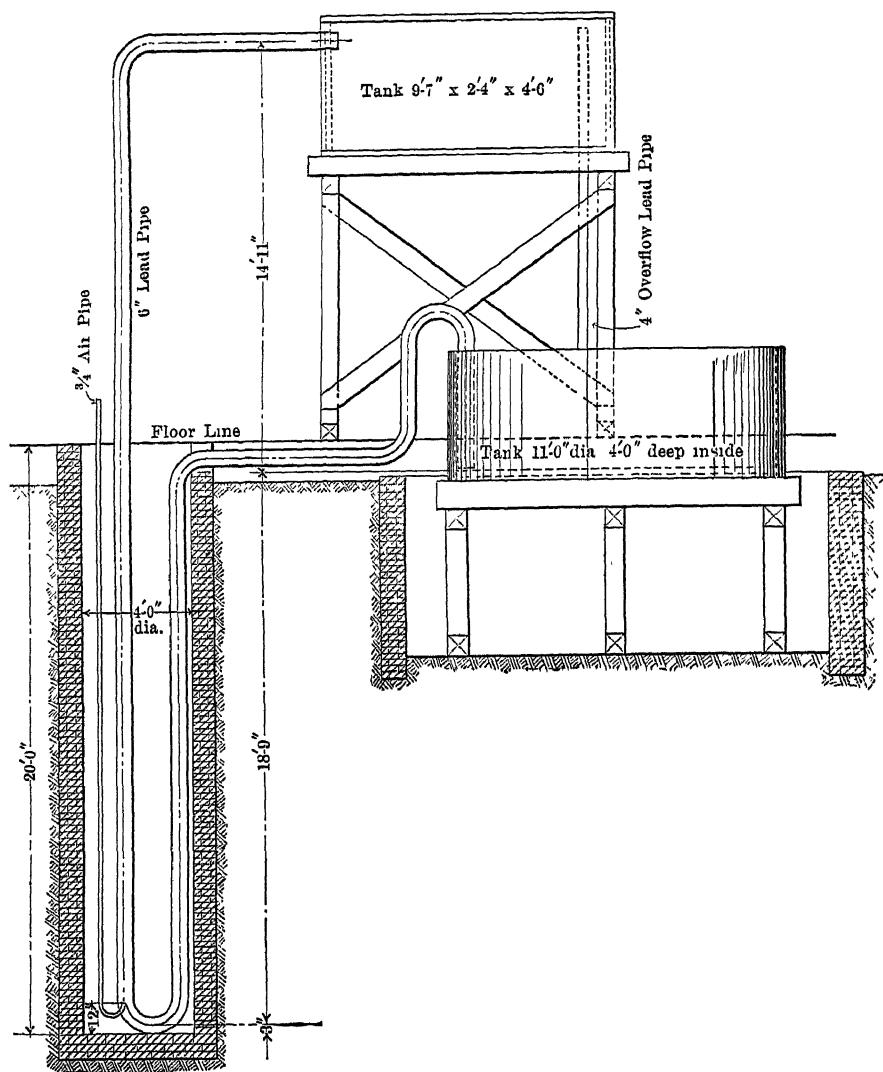
FIG. 8.—ELEVATION OF SOLUTION AIR-LIFT PUMP.

as already described. In the pump room the electrolyte is delivered to
the receiving tank, a round lead-lined tank 11 ft. in diameter and 4.5 ft.
deep (see Fig.. 3). In this tank are 150 ft. of 1-in. 8-lb. antimonial-lead

pipe; live steam is supplied to these coils at a pressure of 65 lb. per square inch, by means of which the electrolyte is heated. From the receiving tank the electrolyte is raised to the discharge tank from which it is again conducted to the head tank of the cascade.

Various means of elevating the electrolyte have been employed in this plant. The first method used was the steam injector constructed of lead and antimony. These pumps were of limited capacity and the dilution of the solution was excessive. They were followed by a system of lead-lined cast-iron eggs in which the electrolyte was collected and blown to

Table VII.—Effect of Speed of Circulation of Electrolyte on the Concentration of the Solution.

Current density 36.0 amperes per square foot, 4-day cathodes, converter anodes. Circulating Electrolyte at Rate of 6 gal. per minute.

Zone	Specific Gravity	Grams per Liter						Copper per Kilo-watt-hour Lb.
		Acid	Cu	As	Sb	Fe	Cl	
1	1.213	174	38 4	7 5	0.56	2 95	0 044	
2	1 216	172	39.2	7.2	0 59	2 95	0.045	
3	1.216	171	40.2	7 1	0.59	2 95	0 043	
4	1.228	153	49.9	7.2	0.59	2 90	0 044	
Average	1 218	168	41.9	7.3	0.58	2.94	0 044	3.92

Circulating Electrolyte at the Rate of 4 gal. per minute.

1	1.211	163	40.0	7 3	0 52	2.90	0 042	
2	1.214	161	41 1	7.3	0.55	2.92	0.040	
3	1.216	159	42.0	7 4	0.58	2 95	0.040	
4	1.263	139	67.8	7.4	0.55	2.90	0 040	
Average	1.226	156	47.7	7.4	0.55	2.92	0.040	3.85

Circulating Electrolyte at rate of 2 gal. per minute.

1	1.210	167	37.8	7.6	0.52	2.90	0.042	
2	1.213	165	39.0	7.1	0.49	2.95	0.042	
3	1.217	162	41.0	7.6	0.51	2.95	0 042	
4	1.265	146	66.4	7.9	0.55	2.95	0.041	
Average	1.226	160	46.1	7.6	0.52	2.94	0.042	3.80

Sampled after circulation had been entirely shut off for 7 hr. with current passing at the rate of 36 amperes per square foot.

1	1.185	179	23.4					
2	1.210	164	38.0					
3	1.230	151	51.8					
4	1.255	138	65.5					
Average	1.220	158	44.7					

the overhead tank with compressed air. This system was very inefficient and gave way to a type of boiler-feed pump, the water end of which was of bronze. These soon became leaky from corrosion and were abandoned. A type of plunger pump with a lead-lined barrel and rubber valve and plunger was next tried. This pump was continued in service for several years with satisfactory results but was replaced 10 years ago by the air lift, which has continued to hold the field. The air lift as used here is constructed of cast lead-antimony pipe 6 in. inside diameter, ¾-in. walls. The pipe is cast in 4-ft. lengths with flanged joints. A brick well 20 ft. deep and 4 ft. diameter, made waterproof, with a cement lining, contains that portion of the lift that is below the floor level. Working against a head of 14 ft. 8 in., 160 gal. of electrolyte (1.22 specific gravity) is raised per minute with a consumption of 80 cu. ft. of free air compressed to 16 lb. per square inch (see Fig. 8).

The lift, while it is of low mechanical efficiency and sensitive to variations in the air pressure, is very satisfactory in its operation, having no moving parts and requiring but little attention. The loss in temperature of the electrolyte during its passage through the air lift from the receiving tank to the discharge tank averages 0.5° C.

The compressed air for the operation of the lift is obtained from the converter-plant blowing engines which are operated by water power. A motor-driven compressor at the tank house supplies air in cases of emergency.

Circulation of the electrolyte is maintained at a speed of 6 gal. per minute, a lower circulation reducing the production per kilowatt-hour while a higher rate has the effect of disturbing too much the settlement of the slime and increasing the silver in cathodes. Table VII shows the effect on the concentration of the electrolyte of varying the speed of circulation. The samples which were taken in the regular refining tank and marked zones 1, 2, 3, and 4 were obtained in the following manner: A glass tube, closed at the top, was lowered to a point 6 in. below the surface of the electrolyte and a sample collected at this point and called zone 1. Zone 2 sample was taken in the same manner at a point 17 in. below the surface; No. 3, 28 in. below the surface and No. 4, 39 in. below the surface. The column of electrolyte in the tanks was 40 in. high, there being 3 in. of slime in the tanks. These figures represent the averages of 104 samples taken during seven consecutive days from 48 refining tanks at 36 amperes per square foot. The samples were all taken between the center pair of electrodes.

These figures show a rather remarkable concentration of the electrolyte as regards copper and acid with negative results regarding impurities in solution and demonstrated that, while operating at 36 amperes per square foot, circulating the electrolyte at the rate of 6 gal. per minute,

delivering the solution at the surface of the tank and discharging from a point near the bottom, fails to maintain the electrolyte in a homogeneous state but is found to be the most economical speed at which to operate.

The arsenic and antimony are the impurities with which we are chiefly concerned in our refining operations, and as their effects on the conductivity of the copper are much alike our practice is to report them combined as arsenic plus antimony in the cathodes, making a separation on the average monthly sample to note the relations. These impurities in the electrolyte and in the anodes are determined separately.

There is no silver in solution in the electrolyte and all of the silver present in the cathodes is deposited mechanically from the slime in suspension; the slime is entrapped on the rough surfaces of the cathodes, the silver contents increasing with the age and roughness of the cathode. The arsenic and antimony are deposited both mechanically from the slime and electrolytically from the electrolyte.

When refining converter anodes, carrying 0.12 per cent. As, at 34 amperes per square foot the arsenic in the electrolyte is not allowed to exceed 7 g. per liter.

The results of an experiment with covered cathodes and converter anodes may be of interest. Three regular cathode starting sheets were placed in separate frames covered with 8-oz. duck in such a manner as to entirely inclose the sheets. These were substituted for three regular cathodes in a commercial-refining tank and after remaining 2.75 days they were removed, sampled and assayed as follows:

Analysis of Electrolyte.

SPECIFIC GRAVITY	GRAMS PER LITER					
	Acid	Cu	As	Sb	Fe	Cl
1.200	152	41.2	7.0	0.44	1.80	0.04

Analysis of Anodes.

Per Cent. Cu	Oz. Ag per Ton	Oz. Au per Ton	Per Cent. As	Per Cent Sb
99.15	39.0	0.25	0.211	0 04

Cathodes Produced.

	Inclosed Cathode	Regular Cathode
Current density, amperes per sq. ft..............	31.6	35 7
Age of cathodes, days........................	2.75	4.0
Ag, oz. per ton................................	0.2	1 2
As + Sb, per cent..............................	0.0036	0.0042

The lower current density of the covered cathodes is due to the increased resistance offered by the covering.

The canvas-covered cathodes were drawn at 2.75 days rather than four days as the canvas was showing signs of failure. The results show that the canvas covering was effective in preventing but little of the silver-bearing slime in suspension from coming in contact with the cathodes but that it had a lesser effect on the arsenic and antimony contents of the deposited copper.

The original method employed at these works of purifying the electrolyte was the manufacture of bluestone. An amount of electrolyte sufficient to maintain the working solutions at the required degree of purity was daily run off from the tank house to the sulphate plant. Here it was heated to the boiling point by means of steam coils and passed through open tanks containing shot copper into which compressed air was blown until all of the free acid was neutralized. After the necessary concentration by boiling the solution was allowed to cool in an open chute where most of the copper crystallized out. The crystals from the chute were redissolved in water, concentrated, and run into the final crystallizing tanks. The crystals from these tanks were washed, dried and packed in barrels for shipment. The mother liquor, after it had become too foul for further use, was run over scrap iron for the recovery of the small amount of copper remaining in solution. After passing over the iron the solution, which contained only the impurities, was wasted.

This was an efficient method of freeing the electrolyte of impurities and would probably still be employed had not the market for the bluestone failed during 1899. In 1897 2,300,000 lb. of bluestone were produced and shipped from our sulphate plant.

The electrolytic method was next employed for purifying the electrolyte. This process consisted in removing from the working solutions each day such an amount of electrolyte as was necessary to maintain the purity of the electrolyte. This solution was passed through four standard refining tanks containing lead anodes and copper or lead cathodes. These tanks were arranged in a cascade the electrolyte flowing from one tank to another. The Cu, As and Sb contents of the solution, as it left the last tank, depended on the speed of flow. With a circulation of 2 liters per minute but a trace of Cu, As and Sb remained. The ampere efficiency of deposit, however, was extremely low at this rate of flow.

Table VIII shows the results obtained while circulating at 4 liters per minute. The percentages of elimination as shown in this table are high while the ampere efficiency is very low, the two lower tanks in the cascade doing but little work. In subsequent experiments the most economical speed of flow was found to be 7 liters per minute with an elimination of 99 per cent. of the copper, 78 per cent. of the arsenic and 91.1

per cent. of the antimony with a total ampere efficiency of about 50 per cent.

The increase in the iron contents of the electrolyte between the inlet of the first and outlet of the last tank, as shown in Table VIII, is due to

TABLE VIII.—*Removal of Copper, Arsenic and Antimony from Electrolyte in Insoluble Anode Tanks.*

Circulation, four liters per minute. Lead anodes, copper cathodes, 9,000 amperes, 31.8 amperes per square foot.

	GRAMS PER LITER					Volts per Tank	Tempera-ture, C.°
	Acid	Cu	Fe	As	Sb		
Inlet tank No. 1...	144	37.060	6.242	3 200	0.463	....	17
Outlet tank No. 1.	184	7.376	6.813	2.240	0.260	2.22	42
Outlet tank No. 2	194	0.504	7.364	0.400	0 061	2 25	57
Outlet tank No. 3.	208	0.088	7.701	0.056	0.038	2.25	64
Outlet tank No. 4.	216	0.048	7.915	0.028	0.028	2.25	65

the concentration of the electrolyte by reason of the heat evolved by the electrolysis. These analyses are corrected on the basis of an unchanged volume of electrolyte in Table IX.

The process thus far described has removed only the Cu, As and Sb from the solution, leaving the Fe, Ni, Bi and Zn. To remove these the electrolyte, as it leaves the insoluble-anode tanks, is run into a lead-lined tank 13 ft. in diameter and 4.5 ft. deep lined with 12-lb. chemical sheet lead and containing 600 ft. of 1-in., 8-lb. chemical-lead pipe. In this tank it is concentrated to 55° Bé., decanted to an open tank 10 ft. long, 4 ft. wide and 3 ft. deep, where it is allowed to stand for four days, during which time the salts of Fe, Ni, Bi and Zn crystallize out leaving a solution of

TABLE IX.—*Corrected Analyses.*

	GRAMS PER LITER					PERCENTAGE ELIMINA-TION OF ORIGINAL AMOUNTS			Ampere Effi-ciency Per Cent.
	Acid	Cu	Fe	As	Sb	Cu	As	Sb	
Inlet tank No. 1...	144	37.060	6.242	3.200	0.4630	......	......	......	......
Outlet tank No. 1..	169	6.760	6.242	2.050	0.2380	81.8	35.9	48.7	71.70
Outlet tank No. 2..	165	0.427	6.242	0.339	0.0517	17.1	53.5	40.2	19 50
Outlet tank No. 3..	169	0.071	6.242	0.045	0.0308	0.9	9.2	4.7	1.68
Outlet tank No. 4..	170	0.038	6.242	0.022	0.0220	0.1	0 7	1.7	0 15
Totals and aver-ages.........	....	......	......	......	....	99.9	99.3	95 3	23.23

approximately the following composition to be returned to the tank room: H_2SO_4, 1,100; As, 1; Sb, 0.2; Fe, 1; Ni, 5.3; and Zn, 1.5 g. per liter.

The Cu, As and Sb deposited in the insoluble-anode tanks are in the form of a black slime, some of the slime adhering to the cathode but the greater portion settling to the bottom of the tanks. An average assay of this slime is given in Table X.

TABLE X.—*Slime from Insoluble-Anode Tanks.*

(Treating elec'rolyte direct from tank room.)

Moisture, per cent.	10 0
Cu, per cent	55.1
SiO$_2$, per cent	1 1
FeO, per cent	0.4
Al$_2$O$_3$, per cent	0 4
CaO, per cent	0 3
S, per cent	4.1
As, per cent	10.3
Sb, per cent	2 5
Ni, per cent	0 35
Zn, per cent	0.32
Ag, oz. per ton	3.4
Au, oz. per ton	0 02

This material was sent to the blast-furnace plant for treatment.

The method of purifying the electrolyte now in use is a modification of the one just described and consists in running off daily from the tank house about 25,000 liters of solution, concentrating this, by boiling, to 48° Bé. From the boiling tank the concentrated solution is run to crystallizing tanks and allowed to stand for four days. At the end of this period 82 per cent. of the copper will have crystallized out. The mother liquor has the following analysis: Acid, 475; Cu, 17.4; As,

TABLE XI.—*Analysis of Insoluble-Anode Tank Slime.*

(Treating mother liquor from crystallizing tanks.)

Moisture, per cent	9.66
Cu, per cent	46.30
SiO$_2$, per cent	0.38
FeO, per cent	1.66
Al$_2$O$_3$, per cent	0 4
CaO, per cent	1.08
S, per cent	5.02
As, per cent	21.48
Sb, per cent	2.28
Ni, per cent	0.35
Zn, per cent	0.32
Ag, oz. per ton	3.61
Au, oz. per ton	0.03

20.2; Sb, 1.1 and Fe, 15.2 g. per liter. This is run to the insoluble-anode tanks where it is treated for the removal of Cu, As and Sb as described above.

Table XI gives an analysis of the slime obtained when using this method.

After treatment in the insoluble-anode tanks the solution is either returned to the tank room or treated, by further concentration and crystallization as described above, for the removal of the Fe, Ni and Zn.

The analysis of slime from the second concentration and precipitation for the removal of Fe, Ni and Zn is as follows: Moisture, 22.1; S, 16.6; Ni, 5.6; Cu, 1.3; Fe, 10.3; Zn, 2.7 per cent.

The advantage of crystallizing the copper from the solution before treatment in the insoluble-anode tanks is that the ampere efficiency of these tanks is greatly increased because of the concentrated solution treated and that the amount of slime produced is reduced by nearly one-half, as follows:

Production of Insoluble-Anode Tank Slime.

	Wet Slime per Month Lb.	As Per Cent.
Treating electrolyte direct from tank room..............	38,333	10.30
Removing 82 per cent. of copper before treating..... ...	21,600	21.48

The slime from the insoluble-anode tanks is sent to the blast furnaces for treatment, the iron slime is washed for the removal of free acid and sent to the blast furnace or wasted if free of copper.

The arsenic liberated from the anodes in the refining tanks finds its way into the following products in the proportions shown: In cathodes, 1.22; in silver slimes, 17.19; and in purification slimes and cathodes, 81.59 per cent. It is therefore necessary to remove from the electrolyte 81.59 per cent. of the arsenic liberated at the anode.

VII. ELECTROLYTIC SLIME.

From a converter anode carrying 40 oz. silver and 0.24 oz. gold per ton, a slime carrying 43.3 per cent. Cu, 5,000 oz. silver and 34 oz. gold per ton is obtained: See Table I for complete analysis.

This slime is removed from the refining tanks once in 60 days in the following manner: A copper jumper, held by the iron studs provided, connects the two ends of tank bars on adjacent tanks and short circuits four tanks, as shown in Fig. 2. The anodes and cathodes are removed

and the slime, 5.5 in. having accumulated, is diluted with water and pumped by means of a bronze steam injector to a round lead-lined tank, 12 ft. diameter, 4 ft. deep, situated 200 ft. distant in the slime-sampling room. Before entering this tank the slime passes through a lead screen with ⅛-in. holes, ½-in. centers. In this tank the slime is washed with hot water to remove the free acid and soluble copper.

From this tank the slime is dropped into a cast-copper, lead-lined egg from which it is forced by compressed air at 100 lb. pressure into a modi-fied Bushnell filter press containing hard-copper plates and rings, 8-oz. duck being used for filter cloth. Thirteen cakes 26 in. in diameter and 1.25 in. thick are made at each charge; the wet weight is 865 lb. per charge at 21 per cent. moisture. From the filter press the slime is placed on steam-jacketed copper-drying tables where the moisture is reduced to 10 per cent. The slime is then crushed with a roller on a cement floor and sampled in 22,000-lb. lots for shipment.

The slime is sampled in the following manner. The 22,000-lb. lot is shoveled into a conical pile, each shovelful being delivered upon the apex of the cone. To insure thorough mixing of the material this operation is repeated twice. The material is then split-shoveled four times; these operations reduce the sample to about 300 lb. From this point the ma-terial is quartered down with repeated crushing to about 130 oz., the laboratory sample.

The slime is then packed in 10-oz. canvas sacks holding 140 lb. each. Four of these sacks are packed in paper-lined wooden boxes in which shape the slime is shipped to an Eastern refinery for treatment. About 560,000 lb. of slime are produced and shipped per year.

VIII.   POWER.

The power for driving the generators, furnishing the current for de-positing, has been at all times furnished by water power developed at the Black Eagle Falls dam. The electrolytic plant is situated 2,000 ft. from the power house.

The original generator equipment consisted of three Thompson-Houston multipolar, shunt-wound, separately excited generators, 200 kw., each delivering 1,000 amperes at 200 volts. This equipment was supplemented, shortly after being installed, by a third machine of the same type and size. These machines were rope driven from a counter shaft operated by water wheels.

The transmission line between the power house and tank house con-sisted of 36 ½-in. copper wires, 18 wires on a side. These wires were sup-ported on 6 by 8 in. cross arms carrying heavy wooden pins and double

petticoat insulators. The cross arms were supported about 10 ft. above the ground by wooden poles.

This wire line was taken down a few years ago and recently it was decided to put the copper into merchantable form by charging it into the wire-bar furnaces with cathodes. A preliminary test to determine the quality of the copper gave the following results: Conductivity, annealed, 98.4 per cent.; As, 0.0072 per cent.; Ag, 5.15 oz. per ton.

This wire was purchased, on the market, in 1892 as high-conductivity
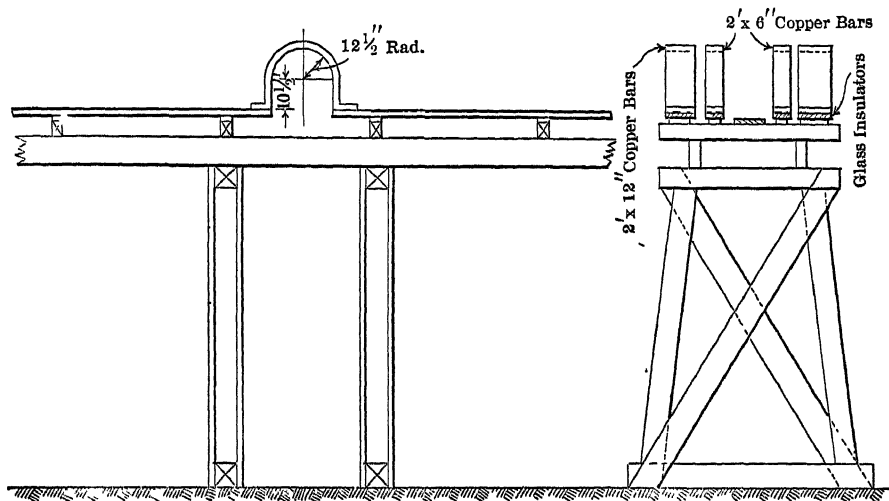
FIG. 9.—SECTION OF TRANSMISSION LINE, SHOWING EXPANSION JOINT.

copper. The consumer of the present day demands copper with a conductivity of 100 per cent. annealed, or over. As previously stated it was decided in 1896 to increase the production by increasing the current density, accordingly, in that year, the four 200-kw. generators were replaced by two Westinghouse, shunt-wound, engine-type, 810-kw. generators having a normal capacity of 4,500 amperes at 180 volts at a speed of 130 rev. per min. These machines were direct connected to a pair of 57-in. horizontal New American turbines operating under a normal head of 41 ft. It may be of interest to record that the average output of these generators to-day is 4,600 amperes each at 222 volts or 1,021 kw., equivalent to 26 per cent. overload.

These generators and water wheels were installed in a new stone power house built to accommodate them and constitutes our power plant of to-day.

To conduct the current from the new generators to the tank house a transmission line of rather unusual proportions was built. This line, as

originally built, consisted of 2 by 12 in. cast-copper slabs in parallel with the ½-in. wire line.

These bars, which were cast in 20-ft. lengths in open iron molds, were supported on a wood trestle (Fig. 9). The joints were made by over-lapping the ends of the slabs 4 in. and fastening them together with three ¾-in. iron bolts, after the surfaces in contact were well cleansed; attempts to solder the joints failed because of the large body of copper and its high heat-conducting property.

Expansion and contraction was taken care of by expansion joints in the form of cast-copper arches. The copper slabs rested on 4 by 4 by 1 in. glass insulators.

Later the wire line was taken down and 4,377 ft. of 2 by 6 in. cast-copper conductors added so that the transmission line now consists of one 2 by 12 in. and one 2 by 6 in. conductor on each side arranged as shown in Fig. 9.

Data on Transmission Line.

Effective length of line:
    2 by 12 in. conductors, both sides......... ....   4,895 ft.
    2 by 6 in. conductors, both sides....... ... .   4,221 ft.

Number of joints:
    2 by 12 in. conductors.........   270
    2 by 6 in. conductors......... ......·.....   234

        Total............... .. -   504

Overlap at junction points:
    2 by 12 in. conductors................................   180 sq. ft.
    2 by 6 in. conductors................... . ...   78 sq. ft.

        Total.......................................   258 sq. ft.

Weight of copper:
    2 by 12 in. conductors....... ... .   466,992 lb.
    2 by 6 in. conductors ....   201,342 lb.

        Total.......................................   668,334 lb.

Number of expansion joints:
    2 by 12 in. conductors   10
    2 by 6 in. conductors...............................   10

        Total...   20

When generator output is 9,295 amperes, 222.1 volts or 2,064.4 kw.
    2 by 12 in. conductors carry.......... ....   6,070 amperes
    2 by 6 in. conductors carry................ ....   3,225 amperes

        Total................ ....   9,295 amperes

Resistance of line at 19° C.:

2 by 12 in. conductors	. . ..	0 002190 ohms
2 by 6 in. conductors ...	. . ..	0 004162 ohms
Combined resistance	. . ..	0.001431 ohms

Temperature rise at 9,295 amperes... ... .   11° C.

Division of resistance:

Due to copper...	75 per cent.
Due to joints	25 per cent.

Power loss:

Drop in line at 9,295 amperes.	13 3 volts
Power loss at 2,064.4 kw	123.6 kw.
Power loss, percentage of total power developed.... ...	5.99 per cent.

The conductivity of the cast-copper bars in this line is approximately 91 per cent. or 9 per cent. less than annealed wire made from the same copper.

When the yearly interest on the investment in this line is compared with the value of the power lost in transmission for a similar period it would appear, from an economical standpoint, that the line contains

Distribution of Energy.

	Volts	Kilowatts	Per Cent. of Total
320 commercial refining tanks in series, 33.4 amperes per sq. ft...................	190 4	1,769.8	85 73
44 starting-sheet tanks, 22 tanks in series, 16.7 amperes per sq. ft...	8.4	78.1	3.78
4 insoluble-anode tanks in series..........	10.0	92.9	4 50
Transmission line........................ ...	13.3	123.6	5 99
Total..........................	222.1	2,064.4	100.0

Analysis of Tank Resistance.

320 commercial refining tanks, 22 anodes, 22 cathodes per tank, 90 per cent ampere efficiency, 2-day cathodes.

	Volts per Tank	Volts 320 Tanks	Per Cent. of 190.4 Volts
Drop between anode busbar and anode	0.044	14.08	7.40
Drop between cathode busbar and cathode	0.055	17.60	9.24
Drop across electrolyte..............	0.496	158.72	83.36
Total..........................	0.595	190.40	100.00

far too much copper, but when the effect on the ultimate cost per ton
to refine is taken into account it is apparent that the investment is fully
justified.

Other power data follow: Generator output, 2-day cathodes, 33.4 am-
peres per square foot; head on water wheels, 44.1 ft.; amperes, 9,295;
volts, 222.1; kilowatts, 2,064.4.

IX.   OPERATING DETAILS.

The main tank room is divided into 10 divisions of 32 tanks each.
Each division is taken care of by two tank men who are held responsible
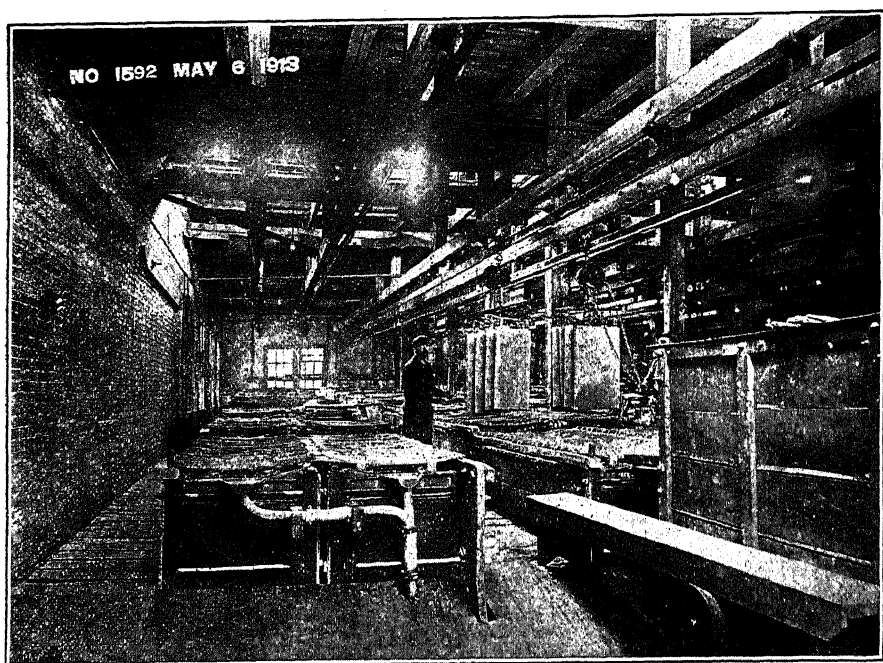
FIG. 10.—DRAWING CATHODES.

for these tanks.   One inspector, locating and removing short circuits
between anodes and cathodes, is employed for each two divisions.   The
inspector's work is done on night shift.

Sixteen tanks of 2-day cathodes, per division, or one-half of the tanks,
are drawn daily, each tank containing 22 anodes and 22 cathodes.   The
day's starting sheets having been delivered to the different divisions by
the night shift, the tank men begin drawing cathodes in the morning.
Four cathodes are drawn at a time with a 1-ton Yale & Towne triplex
chain block and multiple hook as shown in Fig. 10.   The cathodes after

being replaced with starting sheets are deposited in the lead-lined wheel carriages shown in Fig. 11; each carriage holds 66 sheets. The solution dripping from the cathodes is held in these carriages to be later drawn off through an outlet in the bottom. When the carriage is loaded it is moved by hand to the scales, situated as shown in Fig. 3. After weighing the cathodes are lifted from the carriage by the traveling crane. The ends of the angle irons on which the cathode rods rest in the carriage are punched to receive the crane chain hooks by means of which the crane raises the load. After the cathodes are removed, the carriage is weighed
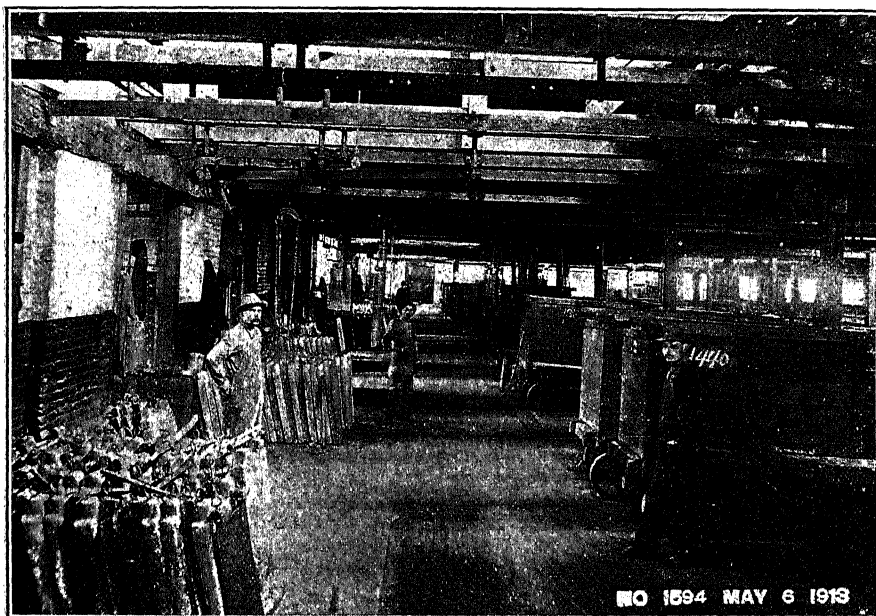
FIG. 11.—VIEW OF WORKING FLOOR.

to obtain the tare. The crane then raises the cathodes and loads them on to a 4-wheel truck for delivery to the furnaces.

After the cathodes are drawn the tank men inspect the anodes, marking such as are ready to be removed as scrap. The same carriages as were used for the cathodes are now spotted under the trolley track at the end of the tanks and the anode scrap is deposited in them. When all of the scrap has been removed in this manner the carriages are moved to the scales and weighed. The crane then lifts the scrap from the carriages, by means of the angle irons, and conveys it to a tank where the slime is washed from the surfaces with a stream of water. The scrap is then placed on trucks, by the crane, from which it is loaded into box cars for shipment to the converters.

After the scrap has been all drawn from the tanks and the carriages removed the tank men proceed to fill the vacancies, thus made in the tanks, with new anodes as shown in Figs. 11 and 12, one anode at a time being handled.

As but four cathodes are removed from one tank at a time it is not necessary to cut the tanks out of circuit while drawing copper.

As compared with the traveling-crane system of handling the anodes and cathodes in tank units, the hand chain-block system of handling, as
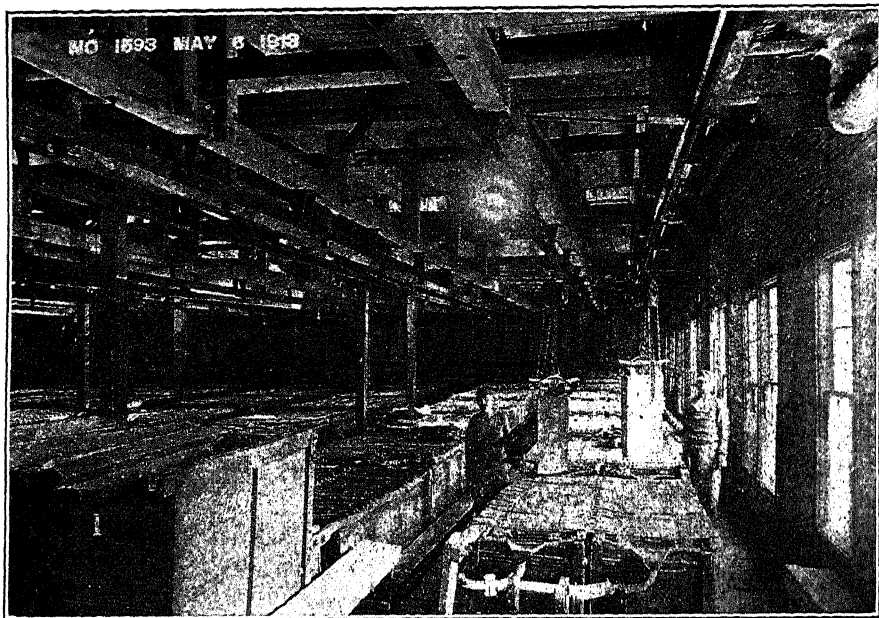
FIG. 12.—CHANGING ANODES.

employed here, admittedly required more labor but is not without its advantages, which are as follows:

First, the chain-block system permits the use of converter anodes.

Second, the chain-block system, with aisles between the refining tanks, permits of starting sheets being straightened and placed in the tanks in such a manner that subsequent removal and straightening is not necessary.

Third, respacing of anodes and cathodes, from time to time, as the anodes corrode, to reduce the resistance of the tank, is unnecessary as the tank men properly space each separate anode and starting sheet as placed in the tank.

Fourth, there is a lower percentage of scrap and no scrap to be returned to the tanks.

Cash prizes are awarded to the tank men on the divisions making the

best and second best records for the month.   The division producing the highest tonnage of cathodes, after deducting the weight of the anode scrap, is considered as having the best record.   This plan has been found very effective in stimulating rivalry between the different divisions and thereby improving the efficiency of operations.

Four insoluble-anode tanks to take care of the increase of copper in solution are provided.   These tanks, as shown in Fig. 3, are situated in a separate room south of the pump house and, by means of a system of circulating pipes, can be run on any of the three electrolytes.   It is not usually found necessary to operate more than two of these tanks at a time.

Four-day cathodes at 34 amperes per square foot are produced in these tanks.   The product of these insoluble-anode tanks is of the same purity as the product of the soluble-anode tanks provided the rate of flow of the electrolyte is sufficiently high to insure ample copper in solution in the lower tanks.   When the rate of flow is greatly reduced or altogether stopped the electrolyte soon becomes impoverished in copper and arsenic is deposited out of the solution with the copper.   The circulation is maintained in these tanks at the rate of 10 gal. per minute.

The following is a summary of the average amount of copper, silver and gold locked up in the process while drawing 2-day cathodes and refining at the rate of 174,000 lb. copper per day.

Metals Locked up in Process.

	Copper, Lb.	Silver, Oz.	Gold, Oz.
In anodes............................	2,300,000	44,000	316
In slime.............................	22,300	140,000	850
In cathodes.........................	180,000	....	....
In solutions........................	95,000	....	....
Totals........................	2,597,300	184,000	1,166

To arrive at the weight of copper in the refining tanks, for inventory purposes on the first of the year, a spring balance is attached to the chain-block hook and the anodes and cathodes are raised clear of their supports and weighed in solution.   A factor, previously determined, representing the difference in weight of the average electrode, weighed in solution in this manner, and its actual weight, is applied to the weights thus obtained. Weights obtained in this manner are found to check actual weights very closely.

The amount of slime in the bottom of the refining tanks at any time is calculated from the tank cleaning record, the number of days the slime

has been accumulating and the amount of slime that is liberated at th anode per tank per day per ampere being known.

Current readings are taken and recorded hourly from a 10,000-amper Weston station ammeter the shunt of which is situated in the center c the tank house. At the power house on each generator-switchboar panel is a 5,000-ampere instrument of the same type. The averages c the hourly readings from these instruments are compared daily wit the readings of the tank-house instrument to detect possible curren leakage.

A system of pressure wires connects the refining tanks and the generator with a voltmeter switchboard situated in the office at the electrolyti plant. Hourly readings of the voltages at generators, at the tank hous and at the different divisions and subdivisions of refining tanks are taker and recorded. In this manner abnormal conditions in any group o refining tanks or any change in resistance of the transmission line ar promptly made known.

The daily report contains, on one sheet, the cathode production for the day and to date for the month, the number and weight of starting sheets used, the pounds of copper deposited per tank per day per ampere ir the different sections, the percentage ampere efficiency of the total pro- duction, the amount and percentage of anode scrap for the day and to date for the month, the average amperes, volts and kilowatts for the day and for the age of the cathodes drawn, the average weight per cathode drawn, the pounds of anodes, cathodes and anode scrap on hand, the pay roll for the day and a digest of the electrolyte men's reports showing amount of solution treated in the purifying plant, character and speed of circulation of electrolyte and other minor data.

The present practice is to draw 3-day cathodes on Mondays and Tues- days, 2-day cathodes on the four days following and none on Sundays, the generators running continuously during the seven days.

Under the head of 22 anodes and 22 cathodes per tank in Table VI, the results obtained are shown in detail.

The average daily production of refined copper is 174,000 lb.

X. FURNACE REFINING.

The refining-furnace plant, to be operated in connection with the electrolytic refinery, began producing anodes from converter copper in October, 1892.

The original installation consisted of two coal-fired reverberatory fur- naces. These furnaces were identical in design, having a hearth 15 ft. 6 in. by 10 ft. 6 in. with a firebox 4 by 4 ft. Each furnace was provided with a separate brick stack 65 ft. high, 28 by 28 in. inside. The capacity of these furnaces was 30,000 lb. of copper each.

The furnace building was 80 by 128 ft. steel frame covered with corrugated iron.

During 1893 and 1894 two additional furnaces of the same type and capacity were added. Two of the furnaces were employed in producing anodes from converter pig and anode scrap from the electrolytic plant and two were used to make wire bar, cake and ingot from the cathodes.

All of the furnaces were dipped by hand, 9-in. ladles being used. The copper was cast in iron molds, the electrolytic copper being dumped in boshes containing water. The anodes were removed from the molds, by means of hooks, and allowed to cool in the air.

Later these four furnaces were replaced by two larger furnaces of an estimated capacity of 50,000 lb. of copper each, subsequently increased to 100,000 lb. each. The hearths of these furnaces are 14 by 24 ft. and the fireboxes are 7 ft. long by 8 ft. wide.   One of these furnaces is connected to one of the original brick stacks, described above, and the other is provided with a new brick stack 74 ft. high with a 34 by 34 in. flue.

These two furnaces with a combined capacity of 200,000 lb. per day now handle the cathode output of the electrolytic refinery, the anodes coming direct from the converters.

The product of the refining furnaces is dipped into copper molds, made at the furnaces. These molds are hung in trunnions over cast-iron boshes through which cold water is circulated. The mold is capsized, as soon as the metal is set, and the copper drops into the cooling water. Cooling in this manner prevents the formation of cupric oxide on the surface of the castings.

The copper is conveyed from the furnaces to the molds by means of large ladles suspended by chains from a trolley running on a $\frac{3}{4}$ by 4 in. track suspended over the boshes. Hand-forged Welsh ladles of the following sizes and capacities are used: 14 in., capacity 200 lb. of copper; 16 in., 250 lb. of copper, and 19 in., 350 lb. of copper. The size of the ladle used is governed by the weight of the casting to be made.

The principal forms into which the copper is cast are as follows: Ingot, ingot bar, cake, wire bar and a special form of round billet used in the manufacture of seamless copper tubing. The copper is shipped direct to the consumer. A 100,000-lb. charge is taken out of each furnace once in 24 hr.

The usual practice of rabbling and poling the copper, for the oxidation of impurities and bringing of the metal to the proper pitch for casting, is carried on. The rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath by means of two $\frac{3}{4}$-in. iron pipes, the end of the pipes being kept below the surface of the metal. About two hours is required for this operation. The re-

ducing action is obtained by forcing green pine poles into the bath and covering the surface of the metal with charcoal. About 28 poles, 8 in. at the butt and 4 in. at the top, and 35 bushels of charcoal are required per charge of 50 tons of copper.

The condition of the bath, as regards oxygen contents, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough-pitch" stage has been reached the poles are removed and the dipping operation is begun.

As the dipping proceeds the refiner observes the set of the castings made and adds charcoal or logs of wood to maintain the oxygen at the proper point.

The skimming of the furnace takes place just before the rabbling or oxidizing is started. The slag obtained is equivalent in weight to about
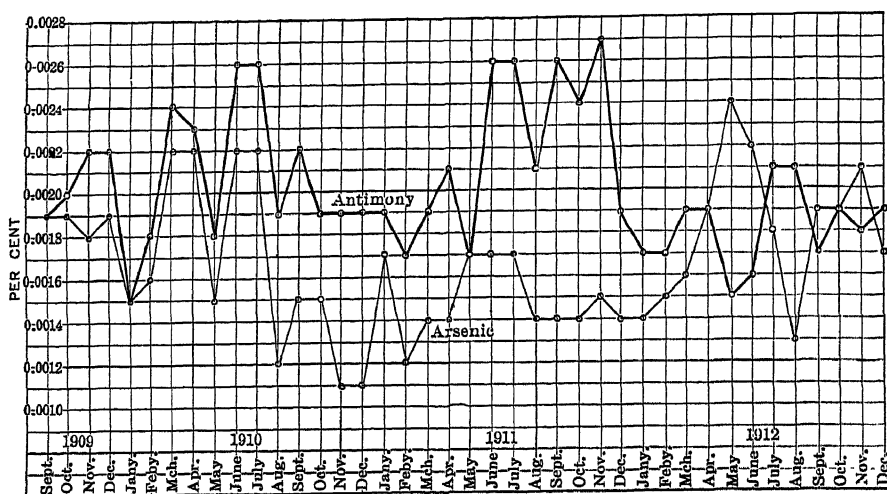
FIG. 13.—RELATION OF ARSENIC TO ANTIMONY IN WIRE BAR.

3.5 per cent. of the copper charged, varying with the amount of lime used. The following is the average analysis of slag produced at furnace No. 2 producing wire bar: Cu, 62.89; SiO_2, 19.1; FeO, 2.0; Al_2O_3, 1.8; CaO, 7.2; MgO, 1.4; S, 0.17; As, 0.02; Sb, 0.025 per cent.; Ag, 0.350 oz. per ton.

The fuel used in these furnaces is known as Lochray lump coal, a bituminous coal mined at Tracy, Mont., 15 miles from these works. The analysis is as follows: Moisture, total, 7.01; moisture, combined, 1.7; volatile matter, 24.7; fixed carbon, 48.0; ash, 18.5; sulphur, 3.5 per cent.; B.t.u. per pound of dry coal, 10,803.

It will be noted that this coal is high in sulphur. To protect the cop-

per as it melts from the sulphur dioxide gases, resulting from the combustion of the fuel, about 30 per cent. of the cathodes going to make up wire bar charges are dipped in milk of lime before charging. As the copper melts, the lime forms a protective covering for the metal, hindering in a large measure the absorption of sulphur by the molten copper. This whitewashing process accounts for the lime and magnesia in the furnace slag.

The average analysis of wire bar produced at these furnaces is shown in Table I, and the relation of arsenic to antimony in wire bar is shown in Fig. 13. The analysis of cake, ingot and other furnace products, in which high electrical conductivity is not essential, will contain 0.02 to 0.05 per cent. less copper, due to the increase in oxygen contents.

XI. PHYSICAL TESTING.

Samples for chemical analysis and physical tests are taken from each furnace charge as follows. A shot sample is made when the furnace is ready to dip. This sample is used for the As + Sb and silver determinations; the month's accumulation of silver buttons, from each furnace, is parted for gold.

Four test bars measuring 3.5 by 3.5 by 11 in. long are cast from each charge of wire bar dipped. The first bar is made when dipping begins, the second when the charge is one-third dipped, the third bar when the charge is two-thirds dipped and the last bar when the last round is being dipped. From each of these bars a billet 1 in. square and 8 in. long is sawed on a band saw; this billet is heated in a Hoskins electrical furnace and rolled down to a ¼-in. rod in an 8 by 13 in. three-high rolling mill. The rod, after being annealed in the electric furnace, is drawn to a No. 12 B. & S. gauge wire on a No. 1 bull block. The rolling mill and the bull block were furnished by the Waterbury Farrel Foundry & Machine Co. of Waterbury, Conn. In drawing the ¼-in. rod to a No. 12 gauge the wire passes successively through the following dies: B. & S. gauge, Nos. 2, 4, 6, 8, 10, and 12. As the wire is not annealed during the process of drawing the product is known commercially as hard-drawn wire.

The No. 12 finishing die is a Waterbury Die Co.'s diamond die which finishes the wire accurately at 0.0808 in. diameter. The preceding dies are ordinary steel dies.

In the physical laboratory these wires are tested for conductivity, tensile strength, elongation and torsion.

The first operation in the laboratory is to anneal the specimen to be tested for conductivity. This is done by suspending the sample between two copper blocks, with 7-ft. centers, and allowing an average current of 124 amperes to pass through the wire for a period of 40 sec. The

initial current is 130 amperes. As the temperature of the wire rises the current decreases until but 118 amperes are passing at the finish. The wire is then allowed to cool in the air. Plunging the wire into water immediately after shutting off the current results in the same conductivity as when cooled in air.

The annealing operation has the effect of increasing the conductivity of the copper about 2.7 per cent. While conductivity tests are made on both the hard-drawn and annealed wire the results on annealed wire are considered far more reliable than those obtained on the hard-drawn wire. This is due to the fact that wire may be drawn to different degrees of hardness by varying the speed at which the wire is drawn and the size and number of dies employed.

Assuming two wires be made from the same bar but drawn to such different degrees of hardness that their conductivities will vary as much as 1 per cent., if these wires are subsequently annealed their conductivities will be found to check very closely.

No. 12 B. & S. gauge wire is used for all physical tests at these works. Conductivity tests are made on both hard-drawn and annealed wire; the tests on hard-drawn wire being made more as a check on the annealed-wire results than for any other value they may have.

The mechanical tests are all made on hard-drawn wire. The physical-testing laboratory building, 15 by 52 ft., was erected and equipped in 1897 and since that date regular tests have been made on all wire-bar charges dipped and for 10 years past conductivity tests have been made on all charges of refined copper.

The first conductivity-testing apparatus installed was designed and furnished us in 1897 by Prof. W. L. Puffer of the Massachusetts Institute of Technology. This bridge required a wire sample 50 ft. long. Accurate results were obtained with the apparatus but its size made it rather unwieldy.

In 1899 the Puffer apparatus was replaced by a Willyoung conductivity bridge furnished by J. G. Biddle of Philadelphia. This bridge handled samples 30 in. long, the conductivity of the samples being calculated from the bridge readings. This instrument was continued in service until 1906 when it gave way to a Hoopes conductivity bridge furnished by the Leeds & Northrup Co. of Philadelphia. This is a most satisfactory instrument. This bridge tests samples 38 in. long and the percentage conductivity is read directly from the scale, no calculation being necessary.

Wire samples are sent from time to time to the Bureau of Standards at Washington. The wires are measured for conductivity by them and returned to us with their certificates of tests. These wires we retain as standards with which we check our conductivity bridge.

The mechanical tests are made on a Riehlé 10,000-lb. horizontal wire-

testing machine and a Riehlé torsional wire-testing machine both furnished by the Riehlé Bros. Testing Machine Co. of Philadelphia.

The first test bar made at the furnace, just as the dipping of the charge is begun, is immediately made into wire and tested for conductivity; about 40 min. is required after the bar is received to obtain the conductivity. With this knowledge the refiner is aided in the subsequent handling of the charge.

Wire is made from the three remaining bars on the following day and the averages of the tests made on the four bars are reported as representing the charge.

The following are the averages of the physical tests made on the wire-bar product at this plant during the year 1912.

```
Conductivity, hard-drawn wire, per cent.......  .. ......    97.32
Conductivity, annealed wire, per cent ..........  .....  ....   100.08
Elongation in 5 ft., per cent...  .....  .....................    1.00
Tensile strength, lb. per sq. in..............  .............. 64,400
Torsion, twists in 6 in. ...................................     37
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NOTE: All mechanical tests made on No. 12 B. & S. gauge hard-drawn wire.

Oxygen in the refined copper is made the subject of a separate paper.[3]

DISCUSSION.

PROF. JOSEPH W. RICHARDS, South Bethlehem, Pa. :—I think Mr. Burns spoke of two unusual conditions which he has at his plant, namely, that he uses directly converter anodes and very high current density. He did not mention another condition: the very low cost of power, which circumstance, I believe, is responsible for some of the peculiarities of his plant noticed by us from the East. I am struck by the rather low ampere efficiency which Mr. Burns obtains, and in looking over the plant thought that there was a considerable leakage from one tank to the other, and that the ampere efficiency could be increased by looking after the leakage of electricity from one tank to the other.

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[3] Notes on the Metallography of Refined Copper, by E. S. Bardwell, p. 742, this volume.

Notes on the Metallography of Refined Copper.

BY EARL S. BARDWELL, GREAT FALLS, MONT.

THE structural relations existing between cuprous oxide and copper were first systematically studied by Heyn[1], who suggested that a study of the microstructure of refined copper might be substituted for the analytical methods in vogue for determining the oxygen content of copper coming from the refining furnace. Hofman, Green and Yerxa,[2] in their paper A Laboratory Study of the Stages in the Refining of Copper, working along the lines suggested by Heyn, showed that a study of the microstructure could be made to yield a correct estimate of the oxygen content of refined copper.

In order to determine the oxygen content of the specimen of refined copper under examination a photomicrograph was first made. From the relative areas of copper and eutectic, the percentage of cuprous oxide and the corresponding percentage of oxygen were then readily calculated. It was found that the values for oxygen thus obtained corresponded closely with the analytical results.

With the idea of utilizing this method of determining the oxygen content of each charge dipped from the refining furnaces at the Boston & Montana Reduction Plant, the physical testing laboratory was equipped with suitable apparatus for carrying out the work. It may be of interest to describe certain of the methods which have been worked out in this connection and give such data as have been obtained bearing on the effect of the oxygen content of refined copper upon its conductivity.

Apparatus.—For the benefit of those who may be unfamiliar with metallographical apparatus and methods and may be interested to know what equipment is required, a few words with regard to the apparatus are in place at this point.

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[1] *Mittheilungen aus den Königlichen Versuchsanstalten zu Berlin,* vol. xviii., p. 315 (1900). See also *The Metallographist,* vol. vi., p. 49 (1903).

[2] *Trans.,* xxxiv., 671 (1903).

The polishing head shown in Fig. 1 is provided with two polishing disks, one on either side. These polishing disks are driven by a direct-current variable-speed motor, direct connected. The optical apparatus shown in Fig. 2 consists of a Bausch & Lomb compound microscope with suitable combinations of eyepieces and objectives, a Bausch & Lomb direct-current self-regulating arc, and a camera stand and camera.

Samples.—From each charge dipped from the refining furnaces two test bars are dipped, one when dipping is started and one at or near the end. In the case of wire-bar charges two intermediate bars are dipped. These test bars have the same cross-section as the wire bars, but are shorter. From these test bars are sawed the samples which are drawn into wire for the physical tests; *i. e.*, tensile strength, torsion, and conductivity. At the same time a piece about 0.5 in. in cross-section and 4 in. in length is sawed out for the metallographical laboratory. A piece about 0.5 in. cube is taken for microscopical examination. One specimen from each charge is examined, except in case of a charge which shows a low conductivity, in which case the last test bar is also examined and the oxygen content determined. In this way we know the best and the worst about every charge dipped from the furnaces and are able to retain a permanent record of each charge in the form of photomicrographs.

Polishing.—The specimen is polished in the usual way. If the polishing has been done carefully the structure can be seen faintly with the naked eye. If upon examination under the microscope the specimen appears satisfactory it may be etched in such a way as to bring out the structure very distinctly.

Etching.—To bring out the structure sufficiently well for the taking of a satisfactory photomicrograph is a difficult matter. A suitable method for etching seemed, therefore, desirable. Nitric acid and ammonia were tried; but while occasionally good results were obtained, neither reagent seemed to be suited to our work. Finally the experiment of heating a polished specimen in a current of dry hydrogen gas was tried. The results obtained were startling. This reagent offers the advantage, unusual among etching reagents, of attacking the cuprous oxide without in any way affecting the copper. The reaction upon which the method depends is $Cu_2O + H_2 \leftrightarrows 2\ Cu + H_2O$. This reaction is reversible, and the exact conditions necessary for obtaining the best results have not as yet been worked out. The indications are that to obtain the best results the specimen should be heated at about 300° C. and allowed to cool in a current of hydrogen. Excellent results have been obtained by heating in this way for 4 or
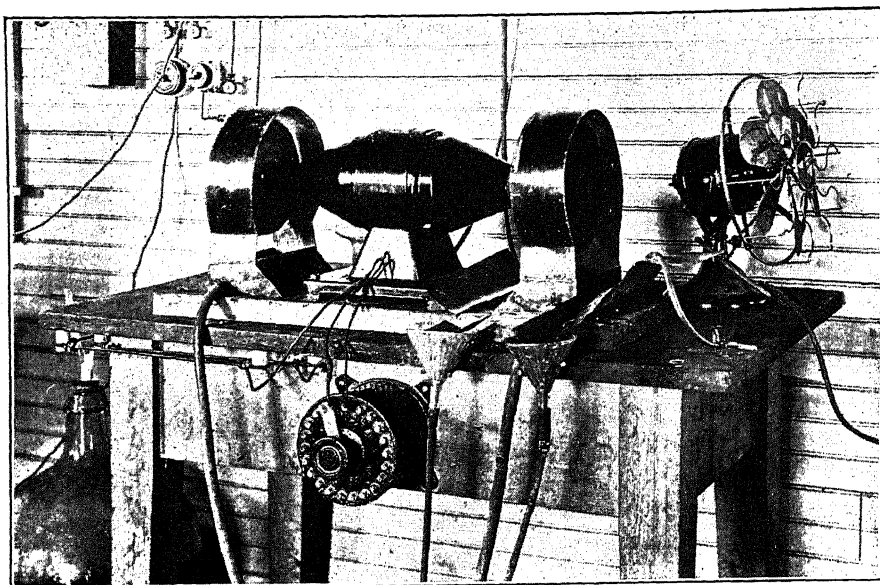
FIG. 1.—POLISHING WHEEL.
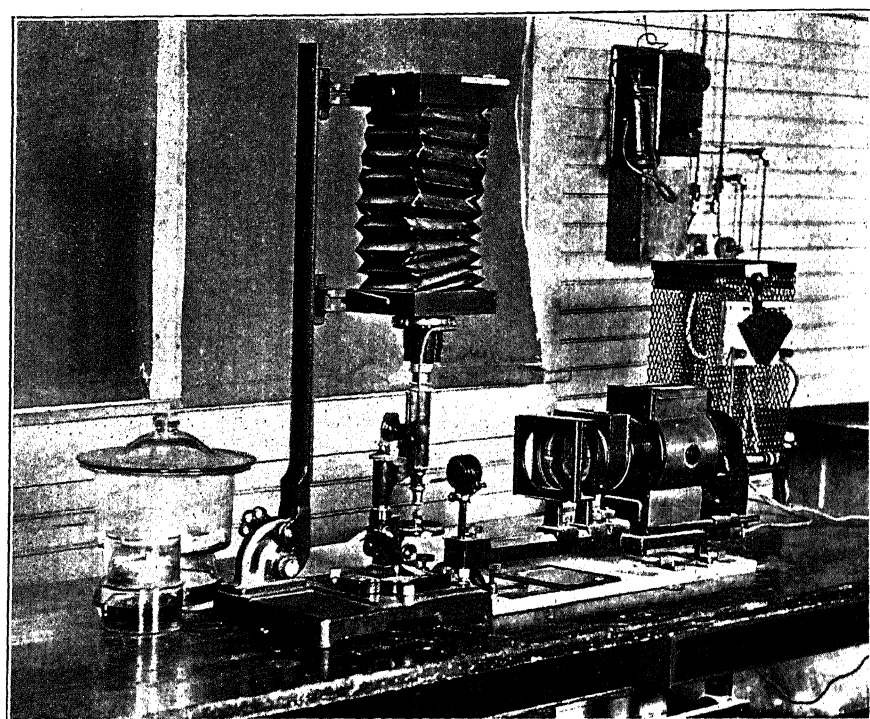
FIG. 2.—METALLOGRAPHICAL APPARATUS.

5 min. Heating to too high a temperature or cooling too suddenly tends to cause cracks in the polished surface, varying from cracks visible to the unaided eye to those visible as fine hair lines under the microscope. The latter are not serious if they are not so numerous as to obscure the structure. The structure in the case of the etched specimen is clearly visible to the naked eye, the network of eutectic appearing as if depressed below the surface of the rest of the specimen, while the copper areas stand out in relief. Under the microscope the eutectic network stands out black on a very nearly white background, and may be photographed with results that are extremely gratifying.

The apparatus required for the etching consists of a Kipp hydrogen generator, a wash bottle containing a concentrated solution of caustic soda, a drying tower containing stick caustic soda, a U-tube containing calcium chloride, a silica tube in which the specimen is placed for heating, and another U-tube containing calcium chloride.

Where the specimen is to be etched with hydrogen it has been found possible to simplify the polishing somewhat, as the etching may be depended upon to bring out the structure. When the specimen shows the structure under the microscope reasonably well it is ready to be etched. The silica tube is disconnected, and the polished specimen, which should be clean and dry, is inserted. The silica tube is then reconnected and hydrogen passed through the apparatus at the rate of 12 to 15 bubbles per minute for 10 min., in order to displace any air that may be contained in the apparatus. The silica tube is then heated by means of a Bunsen flame directed against the tube at the point where the specimen is located. The tube should not be allowed to get hotter than a very low red heat. Heating is continued for about 3 or 4 min., at the end of which time the flame is removed and the tube allowed to cool to room temperature with the hydrogen still passing. The apparatus must not be disconnected until the tube has become quite cool, otherwise there is danger of the specimen becoming oxidized or tarnished. The presence of moisture in the silica tube when the specimen is cooling also seems to cause a tarnish. This is due to the reversible nature of the reaction upon which the operation is dependent, and is obviated by having a sufficiently rapid circulation of dry hydrogen gas to sweep the water resulting from the primary reaction over into the calcium chloride tube.

The specimen upon being removed from the tube will be found to have retained its polish, but the structure will now be clearly visible to the unaided eye. The little cuprous oxide areas in the eutectic have been robbed of their oxygen so as to cause innumerable minute

pits, and it is the shadows which are cast in these pits under reflected light that bring out the detail when examined under the microscope. The eutectic areas thus appear black when illuminated by reflected light, while actually they are no different in color than the rest of the specimen. The polished surface under the microscope is all of one color; *i. e.*, the color of pure copper.

The photomicrograph is now taken in the usual manner, which I do not need to describe in detail.
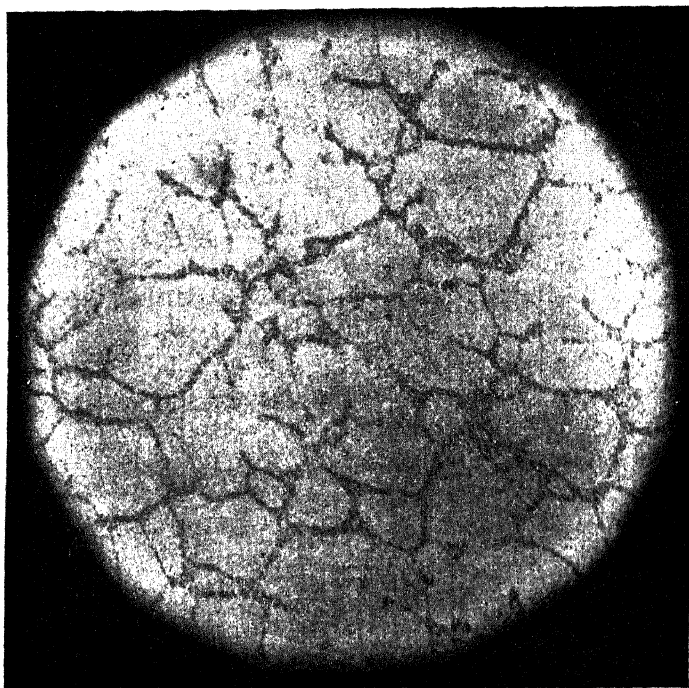
FIG. 3.—REFINED COPPER. UNETCHED. 0.071 PER CENT. OF
OXYGEN. 87 DIAMETERS.

Fig. 3 represents one of the best photomicrographs which we were able to obtain without etching. This specimen contained 0.071 per cent. of oxygen. Fig. 4 shows the photomicrograph of a specimen containing 0.0715 per cent. of oxygen which has been etched by means of hydrogen. Fig. 5 is the photomicrograph of a specimen similarly etched but containing 0.1409 per cent. of oxygen.

Estimation of Oxygen Content.—The measurement of the eutectic after the manner outlined by Hofman, Green, and Yerxa is a tedious process, requiring from 1.5 to 2 hr. Eye estimation, on the other hand, while a valuable aid in controlling the operation of a furnace, is unsatisfactory from the point of view of accuracy and the making

of permanent records.   One can very readily get a relative estimate of the amount of oxygen present in the specimen under examination, but two observers are liable to differ considerably in their respective estimates.   In order to eliminate the personal equation as far as possible, a method that is capable of yielding an accurate estimate is desirable.   In order to be available for practical purposes the method must, moreover, be capable of giving not only fairly accurate results, but results that are quickly obtainable.
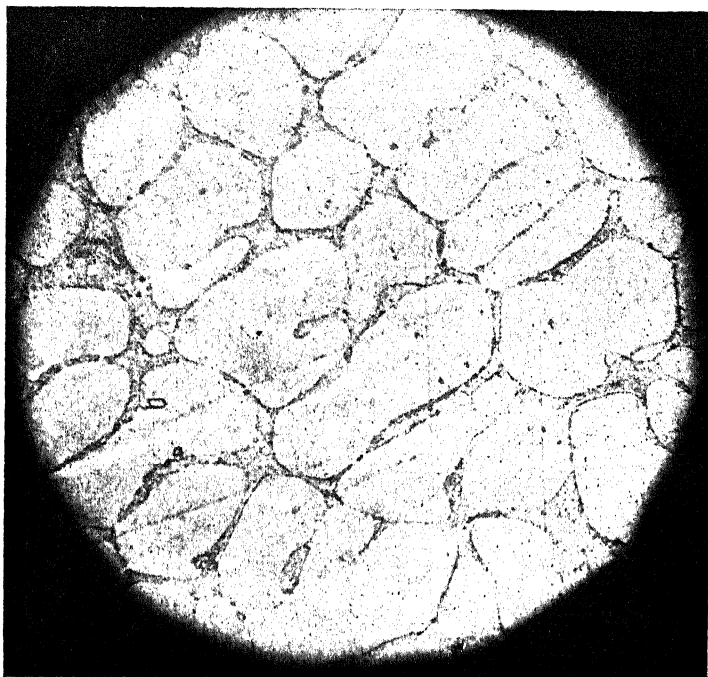
FIG. 4.—REFINED COPPER.   ETCHED BY HYDROGEN.   0.0715 PER CENT. OF OXYGEN.   87 DIAMETERS.

The method at present in use consists in projecting the image, microscopic field, directly upon a piece of "Duplex" paper so as to cover a circle 15 or 16 in. in diameter.   When the image has been sharply focused on the paper the outlines are traced lightly with a hard pencil.   This gives directly an enlarged reproduction of the microscopic field.   The copper areas are next cut out with a sharp knife.   We now have a network of paper representing eutectic and some odd shaped pieces of paper representing copper.   The two lots of paper are then weighed on chemical balances, and as the weights obtained are directly proportional to the areas, we have the equivalent of the planimetric method, with very little expenditure of

time. The oxygen is then computed in the same manner as that outlined by Hofman, Green, and Yerxa. Working in this way, it is possible to make four or five determinations in an hour. Separate determinations made on the same test bar will ordinarily be found to give results checking within 6 or 7 per cent.

The experiment was tried of taking a button sample at the same time that the test bar was dipped, and determining the oxygen in both the button sample and the test bar. Two instances of this sort gave results as follows:
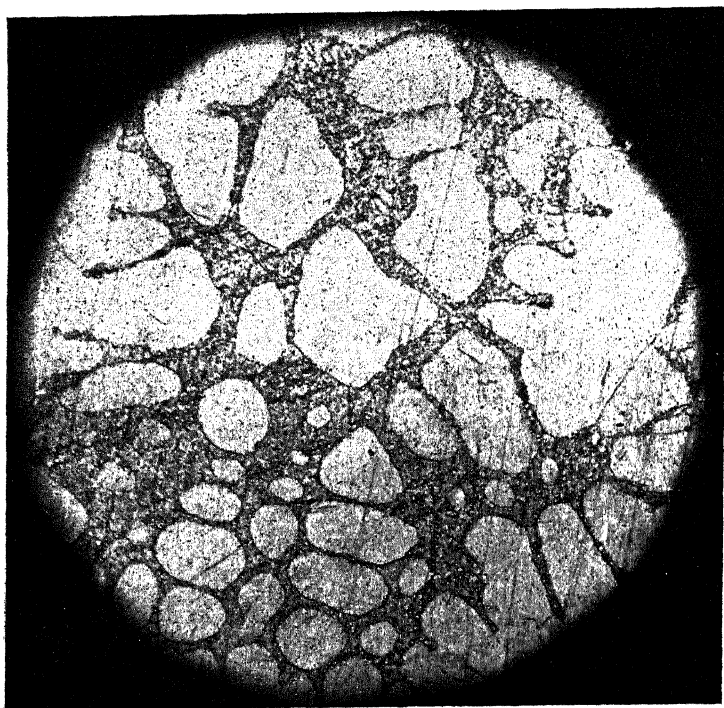
FIG. 5.—REFINED COPPER. ETCHED BY HYDROGEN. 0.1409 PER CENT. OF OXYGEN. 87 DIAMETERS.

Oxygen Content by Measurement Method.

Charge No.	Button Sample. Per Cent.	Test Bar. Per Cent.
2–132–4	0.071	0.076
2–133–4	0.092	0.087

Crude as the method appears at first sight, it is fully as accurate as the planimetric method and is infinitely simpler to work than the standard analytical method. Two examples will serve as a comparison between the results obtained by the measurement method and the results obtained by the usual hydrogen combustion method.

Oxygen Content.

Charge No.	Measurement. Per Cent.	Hydrogen Combustion. Per Cent.
2–192–2	0.0446	0.0492
2–188–2	0.0715	0.0675

A photomicrograph of the charge numbered 2–188–2 is given in Fig. 4.

The hydrogen combustion method for determining oxygen in refined copper is a difference method and is liable to all the inaccuracies of methods of that type. The measurement method is a direct method and should yield more accurate results.
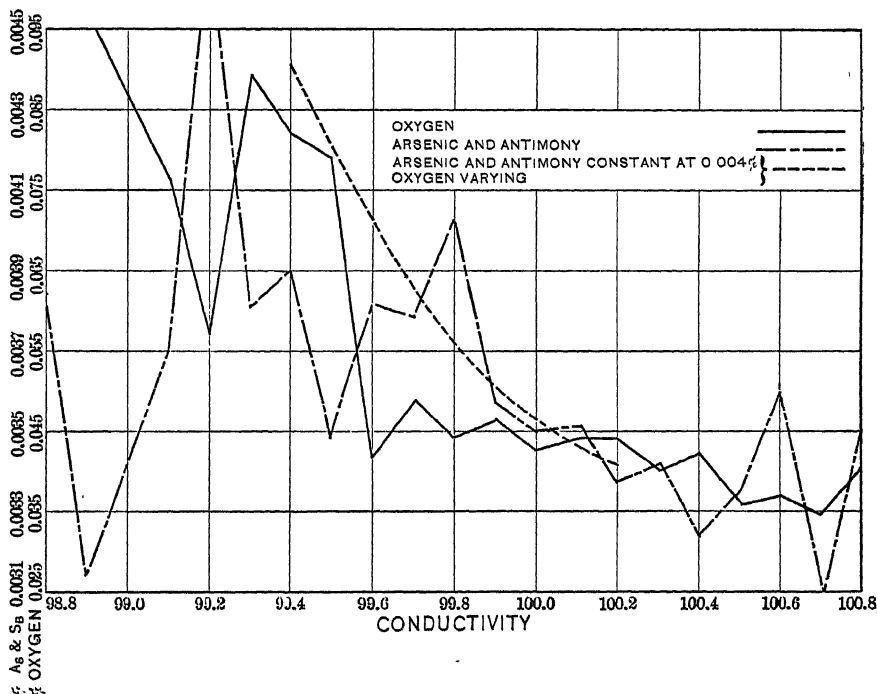
FIG. 6.—CONDUCTIVITY CURVE OF REFINED COPPER.

The image may be projected on the paper in either of two ways. First, with a sufficiently powerful arc the image may be projected directly through the microscope eyepiece. This necessitates special arrangements for darkening the room and for shielding the screen from light reflected from the arc. Second, the photographic negative may be used as a lantern slide and the image projected through it upon the screen. The latter method is the one usually adopted. The room need not be darkened, aside from pulling down shades in the portion of the room where the work is going on.

Effect of Oxygen on the Conductivity of Refined Copper.—By means of the measurement method the oxygen has been determined in about 300 furnace charges, and as we know the conductivity corresponding to each oxygen determination, we are able to present data which are of considerable interest. The curve, Fig. 6, is plotted with conductivities as abscissæ and oxygen content as ordinates. The content in combined arsenic and antimony is here similarly plotted against the conductivity. In making this curve the oxygen content of all samples having the same conductivity has been averaged, and similarly the arsenic and antimony content. The total number of results entering into each average is shown in the following table.

Conductivity. Per Cent.	Number of Determinations	
	Oxygen.	Arsenic and Antimony.
98.9	3	3
99.0	0	0
99.1	2	2
99.2	2	2
99.3	8	8
99.4	7	7
99.5	9	9
99.6	4	4
99.7	10	10
99.8	14	14
99.9	21	21
100.0	31	31
100.1	33	33
100.2	· 35	36
100.3	44	43
100.4	32	32
100.5	23	22
100.6	12	12
100.7	3	3
100.8	3	3

Looking at the curve, we see that while a decrease in oxygen content coincides with an increase in conductivity, the curve is very irregular. It is evident that the conductivity depends upon the combined effect of the impurities and not on the oxygen content alone. As the percentages of the several impurities vary with each charge, some method of studying the effect of oxygen alone seemed to be required. While attempting to ascertain the effect of casting temperature on the physical properties of refined copper a ladleful of the molten metal was held until a portion of the metal had solidified, accompanied, naturally, by an increase of the oxygen content of the mother metal. It was found, however, that the other impurities in the mother metal remained constant, the oxygen alone varying. Thus the phenomenon of selective freezing seemed to offer a method of ap-

proaching the problem. Pressure of other work has thus far prevented following this line out. I shall, however, venture to present the results that were obtained, in hope that they may prove suggestive to some one working along similar lines.

A ladleful of metal was dipped from the furnace and a test bar cast from it. After casting this first bar the ladle was allowed to stand for about 1.5 min., at the end of which time a second bar was cast. After 1.75 min. a third bar was cast. Photomicrographs of specimens from each of these bars, 1, 2, and 3, are shown in Figs. 7, 8, and 9. In the photomicrographs the gradual increase in oxygen will be noted. The following table gives complete data as to physical tests and chemical composition:
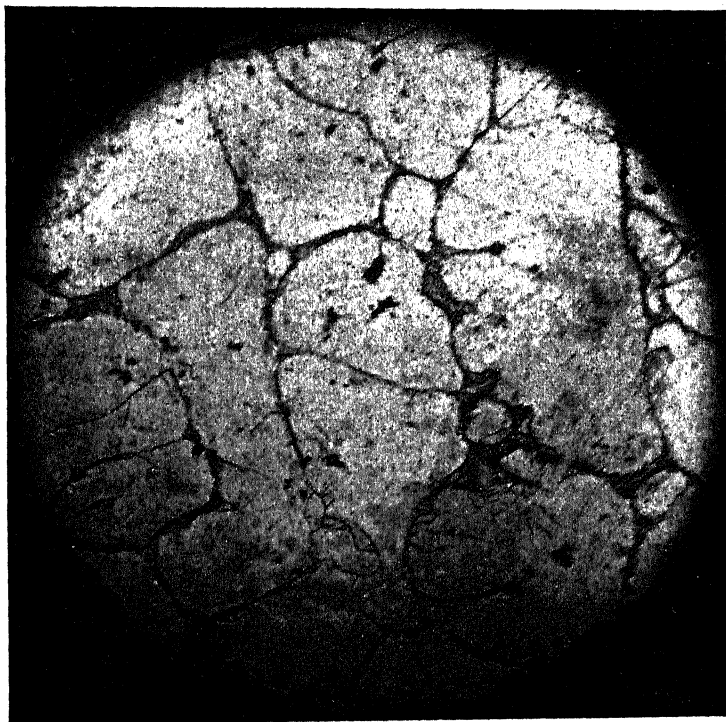
FIG. 7.—BAR No. 1.

Physical Properties.

	Bar 1.	Bar 2.	Bar 3.
Tensile strength, lb. per sq. in.....	66,280	65,540	65,880
Per cent. elongation in 5 ft..........	1.2	1.0	1.1
Conductivity :			
Hard drawn........,....................	97.2	96.8	96.4
Annealed...............................	100.2	99.8	99.4

Chemical Composition.

Arsenic and antimony, per cent...	0.0043	0.0037	0.0036
Silver, ounces per ton................	1.00	1.00	1.00
Cuprous oxide, per cent..............	0.365	0.495	0.815
Equivalent oxygen, per cent.......	0.0408	0.0555	0.0912

The question will probably arise as to how much of the increase in oxygen is caused by the oxygen in the atmosphere. Owing to the considerable amount of metal in the ladle and the fact that the metal was freezing, I shall assume that atmospheric oxygen played a negligible part in the experiment.
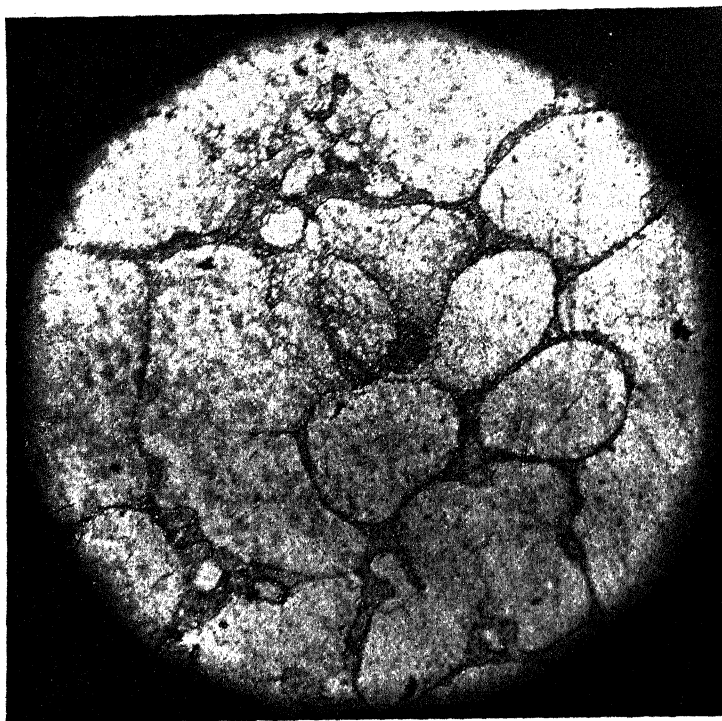
FIG. 8.—BAR No. 2.

Fig. 10 gives the freezing point curve of copper-cuprous oxide alloys plotted from data obtained by Heyn in 1900. Only a portion of the curve is shown, as alloys containing more than 8 per cent. of cuprous oxide are of no practical importance and do not concern us here. The line $a\ a'$, Fig. 10, represents the cooling of Bar 1 (see Fig. 7). When temperature a' is reached copper separates out, and the composition of the mother metal varies along the branch of the curve marked "Freezing Point of Copper" until it reaches a composition represented by 3.45 per cent. of cuprous oxide, whereupon it

solidifies as the eutectic mixture of cuprous oxide and copper, the dark network shown in the photomicrographs. Now, very shortly after the first bar was dipped the metal in the ladle reached the temperature at which it began to freeze in a like manner to that described above in the case of the test bar.

At the time of dipping the second bar the mother metal in the ladle had reached a composition represented by b' b. The oxygen in the second bar is 0.0555 per cent., as against 0.0408 per cent. in the case of the first bar, and the conductivity has dropped from 100.2 to
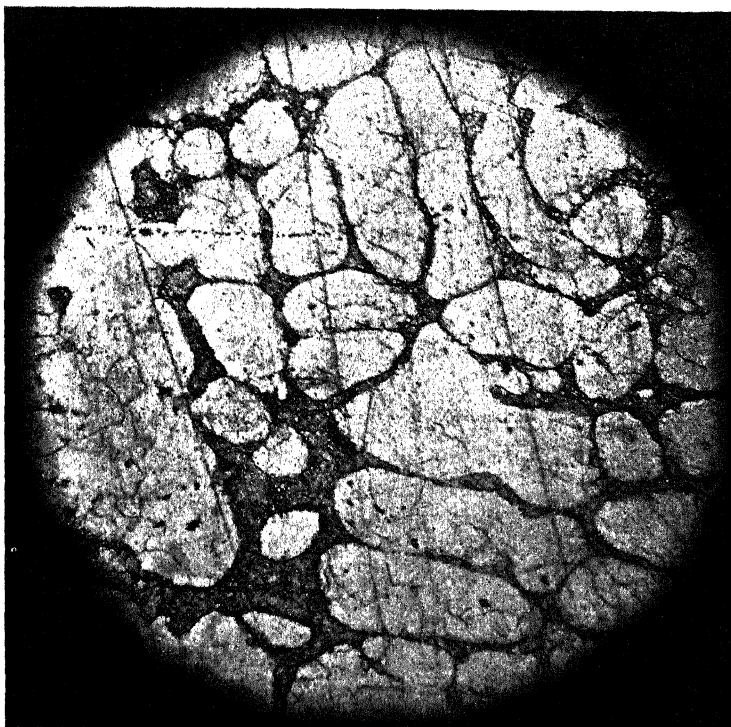
FIG. 9.—BAR No. 3.

99.8 per cent. The mother metal in the ladle continued to freeze selectively until finally when the mother metal had reached the composition c' c the third bar was dipped. This bar contained 0.0912 per cent. of oxygen and had a conductivity of 99.4 per cent. The silver, as will be noted, remains the same in each of the three bars. This is probably due to the fact that silver is slightly soluble in copper in the solid state and is capable of forming mixed crystals with copper to a limited extent. Under the conditions of the test silver would, therefore, separate out with the copper. The solubility of

silver in copper is evidently such that the mother metal becomes neither richer nor poorer in silver as the metal freezes. The arsenic and antimony varies from 0.0043 per cent. in the first bar to 0.0036 per cent. in the third. The accuracy of this determination is such that the arsenic and antimony might be said to be the same in each of the three bars. This again seems to be due to the fact that the compounds which arsenic and antimony form with copper are to a certain extent soluble in copper in the solid condition. With the other impurities remaining constant, or practically so, the falling off in conductivity as we pass from the first bar to the third seems to be due to oxygen alone.
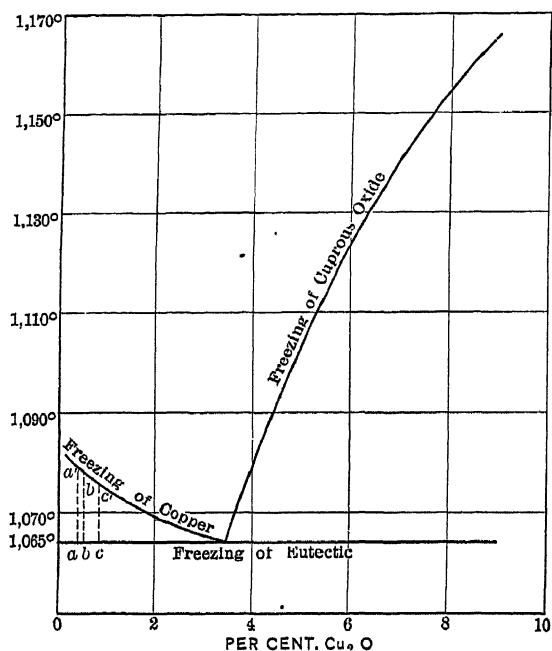
FIG. 10.—FREEZING-POINT CURVE OF COPPER-CUPROUS OXIDE ALLOYS.

These results have been plotted on the curve Fig. 6 and, as will be seen, the points lie along a smooth curve. By comparing a series of tests of this kind made upon different furnace charges with varying amounts of impurities it should be possible to obtain considerable information concerning the effect of oxygen on conductivity in the presence of varying amounts of other impurities, as well as perhaps valuable information concerning the interrelation of oxygen and the other common impurities with regard to their effect on conductivity.

DISCUSSION.

H. O. HOFMAN, Boston, Mass. (communication to the Secretary*):— It is pleasing to the originators of the planimetric determination of oxygen in refined copper to see that the method has been taken up by a large smelting plant to be used in every-day work after it has been simplified to suit the new conditions. Several years ago Huntington and Desch [1] had improved our mode of working. They projected the developed negative on to a screen, traced the outlines of the picture with a pencil, drew a border, and measured one constituent with the planimeter. In order to give the work greater precision, they divided the area into squares of 1 cm. side, shaded the areas of one constituent, and estimated the proportions of shaded and unshaded areas. Mr. Bardwell's plan is simplicity itself, and the data given show that the results are all that can be expected.

The diagram, Fig. 6, showing the electric conductivity curve of refined copper as influenced by the combined presence of oxygen and arsenic plus antimony, shows how important it is to know something about the amount of oxygen present if we are to understand the effects other constituents may have which are present with the oxygen. The requirement for electrolytic copper advocated by Addicks,[2] that the electrolytic conductivity be satisfactory and the metal form a mechanically perfect casting, serves some purposes, but not all. The influence of oxygen upon the mechanical properties of copper which has not passed through an electrolytic plant is marked. It was studied by Hampe [3] in his classical researches, and was emphasized from the point of view of the copper smelter by Lewis,[4] who recently [5] reaffirmed his views in his protest against the specifications of the American Society for Testing Materials [6] of making electric conductivity the basis of a specification for copper which is not to be used for electrical purposes. Extended studies upon the influence of oxygen upon the limits of the harmful effects of impurities, such as arsenic, anti-

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* Received Aug. 26, 1913.
[1] *Transactions of the Faraday Society*, vol. iv., p. 51 (1908).
[2] *Trans.*, xxxiv., 985 (1903).
[3] *Zeitschrift für das Berg-Hütten- und Salinenwesen im preussischen Staate*, vol. xxi., p. 218 (1873) ; vol. xxii., p. 93 (1874) ; vol. xxiv., p. 6 (1876).
[4] *Engineering*, vol. lxxvi., p. 753 (Dec. 4, 1903) ; *Engineering and Mining Journal*, vol. lxxvii., No. 7, p. 284 (Feb. 18, 1904).
[5] *Metallurgical and Chemical Engineering*, vol. x., No. 9, p. 540 (Sept., 1912).
[6] Year Book, 1911, p. 127.

mony, tin, and bismuth, upon the mechanical properties have been made by Johnson,[7] Greaves,[8] Archbutt,[9] Baucke,[10] and others.

The study of the influence of the combined presence of oxygen and impurities upon the electric conductivity of copper has been begun by Bardwell, who unfortunately has had to drop his investigation. A great deal of very accurate physical and chemical laboratory work will have to be done before a satisfactory answer can be given to this important question.

[7] *Journal of the Institute of Metals*, vol. iv., p. 163 (1910) ; vol. viii., p. 192 (1912).

[8] *Idem*, vol. vii., p. 218 (1912).

[9] *Idem*, vol. vii., p. 262 (1912).

[10] *Internationale Zeitschrift für Metallographie*, vol. iii., No. 3, p. 195 (Dec., 1912).

The Determination of Arsenic and Antimony in Converter and Electrolytic Copper.

BY E. E. BROWNSON, BUTTE, MONT.

(Butte Meeting, August, 1913)

THIS paper will be confined to the treatment of methods as applied in the laboratory of the Boston and Montana Reduction Department of the Anaconda Copper Mining Co. at Great Falls, Mont.

As the electrolytic plant of this company operates at a current density of approximately 35 to 40 amperes per square foot when running up to capacity, the arsenic and antimony content of the anodes delivered to the plant is of considerable importance as bearing on the purity of the refined cathodes turned out.

Owing to the rather high relative percentage of antimony, it has always been the custom to determine antimony as well as arsenic in all forms of copper. Thus, anodes have been analyzed in which the antimony was about equal to the arsenic, although the average shows about 2 parts of arsenic to 1 of antimony. In the refined copper the relative proportions of the two impurities vary considerably, but probably in the average are about equal. In this latter case, the percentage of the two is so small that ordinarily no attempt is made to separate them, the report of analysis showing percentage of As and Sb combined.

Although rapid and exceedingly accurate determinations of arsenic may be made by means of various distillation schemes, the determination of antimony, either by itself or together with arsenic, has always been more or less cumbersome and has required a considerable amount of time. By the methods, details of which follow, the determination of arsenic and antimony, either separated as in the case of anodes or together in case of refined copper, is accomplished in 24 hr. from the time the sample is received by the chemist.

As to the accuracy of the results obtained, it can be quite positively stated that in this respect these methods are very satisfactory, much more so than any heretofore employed in our work, and when the rapidity of determination is considered, the results are very accurate.

Arsenic and Antimony in Converter Copper.

Treat 10 g. of the sample in a No. 5 beaker with 30 cc. of sulphuric acid, 20 cc. of nitric acid, and about 50 cc. of water. When solution is complete, heat to boiling to drive off all red fumes, add water to fill beaker about two-thirds and electrolyze about 2.5 hr. at 4 amperes.

The copper is deposited on a large platinum gauze cylinder cathode having the following dimensions: height, 3 in.; diameter, 3 in. The gauze is made of wire 0.0085 in. in diameter and has 36 meshes to the inch. To strengthen the cathode, supporting rings of platinum wire 0.005 in. in diameter are provided at top and bottom. The precipitation is continued until the solution contains about 0.25 g. of copper. It then has a faint blue color. A little experience teaches the proper point at which to remove from the battery. The cathode is rinsed with a little water from the jet when removed from the solution.

The acids are now neutralized with ammonium hydroxide and the solution made acid with hydrochloric acid with 2 or 3 cc. of the acid in excess. Pass a rapid stream of hydrogen sulphide about 30 min. and allow the precipitated sulphides of copper, arsenic, antimony, etc., to settle about 30 min. Filter through a 12.5-cm. paper (C. S. & S. No. 597) in a 3-in. smooth funnel. Discard filtrate. Wash precipitate once on paper with water and then wash sulphides into No. 3 beaker with very smallest possible amount of water. Instead of No. 3 Griffin shaped beaker it is much better to use a No. 5 tall, usual form beaker, as the subsequent evaporation may be carried out much faster without loss by spattering. The beaker containing the sulphides is placed under the funnel and the small amount of sulphides still adhering to the filter is dissolved with 25 or 30 cc. of hot aqua regia poured carefully around the edges of the paper. It is usually well to pour about 10 cc. of this aqua regia into the original No. 5 beaker and transfer from there to the paper, as sometimes a film of antimony forms on the bottom of this beaker In order to prevent too great action of the aqua regia on the filter paper it may be diluted about one-quarter with water. The whole must, however, be very hot to attack sulphides on paper. Wash No. 5 beaker and paper with a small amount of water. At this point it is of great importance to keep the solution as low as possible, consistent with good work, as any unnecessary solution means extra time to evaporate.

Cover the beaker containing the sulphides and aqua regia, boil about 25 or 30 min. to thoroughly break up the sulphides, remove the cover after washing with a small amount of water from a fine jet

and rinse down the sides of the beaker with the fine jet. This should add very little to the bulk of the solution in the beaker. Continue evaporation to complete dryness at a temperature just high enough not to cause loss by spattering. The residue should smell ". sweet."

Add about 4 or 5 g. of stick potassium hydroxide and 30 cc. of water and boil vigorously for 10 or 15 min. All arsenic and antimony will go into solution. Add 25 cc. of strong sodium sulphide solution (1 lb. of fresh mono-crystals to 2 liters of water) and again boil vigorously 5 or 10 min. Allow to cool, decant the clear liquid through an 11-cm. paper (C. S. & S. No. 597) into a No. 2 beaker, again add 25 cc. of the strong sodium sulphide solution to the black sulphides left in the beaker, stir well and transfer to filter paper, washing well with a jet of dilute sodium sulphide (1 lb. to 8 liters of water) from wash bottle. If the operator be sufficiently careful to keep the sulphides well wet on the paper, the above washing may be more conveniently accomplished with a jet of hot water. However, if the copper sulphide be allowed to dry, washing with water is liable to carry through some copper. A convenient bulk for the filtrate is from 150 to 175 cc. Add 5 cc. of good hydrogen peroxide and heat until the strong yellow color of the solution fades. Unless too much organic matter has been taken up by the action of too concentrated aqua regia on the filter paper, the solution should become nearly colorless. Cool and electrolyze over night at from 0.10 to 0.15 ampere. Antimony alone is precipitated.

In the morning remove the cathode, carefully washing the adhering solution which contains arsenic into a clean No. 4 beaker with a jet of water, wash in water and two changes of alcohol, dry carefully over an alcohol flame, and weigh for antimony. When the current is interrupted, the antimony coated cathode should be removed, washed, and dried with as much dispatch as possible, as the antimony is quite soluble in the sodium sulphide solution. The cathode employed for this deposition is an ordinary split, foil cylinder, one that has been roughened by long use.

Transfer the solution from the No. 2 beaker to the No. 4 beaker containing washings from the cathode. Make distinctly acid, using dilute sulphuric acid (1 part of H_2SO_4 to 4 parts of water). Pass a strong stream of hydrogen sulphide about 10 min. Allow to settle 20 or 25 min., and filter through 11-cm. paper. The arsenic is retained on the paper, finely divided sulphur, only, passing through. It is best to decant the partly clear liquid, only, through the paper. Transfer the heavy arsenic sulphide and sulphur with any accompanying solution directly to a No. 2 beaker. This is done with a jet of

water. Now decant any excess water from the No. 2 beaker through paper and place this No. 2 beaker, which contains the arsenic sulphide, under the funnel. Dissolve the small amount of arsenic sulphide on the paper through into the beaker with dilute ammonium hydroxide (1 part concentrated NH_4OH to 4 parts of water. It is best to wash down the sides of the No. 4 beaker in which the sulphides were precipitated with 10 or 15 cc. of this dilute ammonium hydroxide in order to recover any arsenic that may have adhered to the beaker. Then pour this washing from the No. 4 beaker around the edges of the filter paper, thus carrying any arsenic on the paper down into the bulk of the arsenic in the No. 2 beaker. Wash the No. 4 beaker and the paper with a small amount of water from the jet.

Make the contents of the No. 2 beaker acid with sulphuric acid and add 7 or 8 cc. in excess. Evaporate to sulphuric fumes at a temperature just high enough not to cause loss by spattering and then place on very high heat from 1 to 1.5 hr. Cool, wash the rim and the sides of the beaker down thoroughly with water, add water to half fill the beaker, neutralize the acid with ammonium hydroxide, make slightly but distinctly acid with hydrochloric acid, and filter through an 11-cm. paper into a No. 4 beaker, washing thoroughly with hot water. Add water to about half fill the beaker, neutralize the acid with sodium bicarbonate and add 3 or 4 g. excess. Cool to the temperature of the room, add starch solution, and titrate with standard iodine solution.

Titrating solutions :

Starch Solution.—Make a thick emulsion of starch in 30 cc. of water by hard boiling and dilute to 200 cc. Cool, and use as desired. This solution should be made fresh each day.

Iodine Solution.—Dissolve 10.21 g. of iodine in 8 or 10 cc. of water with 17 g. of potassium iodide and dilute to 2 liters. Each cubic centimeter of iodine solution is equal to about 0.0015 g. of arsenic.

To standardize the iodine solution, dissolve 50 mg. of arsenic trioxide in about 30 cc. of water and 1 g. of potassium hydroxide, make acid with hydrochloric acid, alkaline with sodium bicarbonate, add 3 or 4 g. excess of the sodium bicarbonate, and titrate with the iodine solution after adding starch indicator.

In employing the above method for determining arsenic in converter copper, the arsenic should, in order to work most satisfactorily,

be not lower than 0.04 or 0.05 per cent. When the arsenic is present in lower amounts it is well to use a charge of 20 g., in which case the original solution should be made in a correspondingly larger amount of the sulphuric nitric acid mixture, that is, from 90 to 100 cc. This mixture is made up in the proportion of 200 cc. of nitric to 300 cc. of sulphuric acid.

Arsenic and Antimony in Refined Copper.

The lack of a short, satisfactory method for this determination has long been a source of much lost time and of delay to the management of this company. By the following scheme, results which are exceedingly reliable may be turned out in a very short time as compared to methods heretofore employed.

Dissolve 25 g. of the sample and 50 mg. of pure iron in a No. 6 beaker with from 85 to 90 cc. of nitric acid and about 75 cc. of water. When the more violent action ceases, heat gently until the copper is completely dissolved and then boil until all red fumes are expelled. Dilute to about 600 or 700 cc. with very hot water and make ammoniacal. The ammonium hydroxide should be added, with constant stirring, until all copper hydroxide is redissolved. The beaker should be well filled at this point, if necessary by the addition of hot water. Stir thoroughly to collect the ferric hydroxide. Boiling is not allowable, as the copper in ammoniacal solution is too liable to be oxidized. Allow to settle 10 or 15 min., this depending on the time at the disposal of the operator.

Filter through a 12.5-cm. paper, preferably No. 86 Dreverhoff, as this is a very fast paper, admirably adapted for filtering off ferric hydroxide. In this filtration there is a tendency for copper hydroxide to form as the solution becomes cold or is washed with too cold water. A little concentrated ammonia will redissolve this hydroxide at any time it may appear. In the event of any of the ferric hydroxide adhering to the sides of the beaker, it may be conveniently washed down with a jet of dilute hydrochloric acid, 1 part of acid to 1 of water, or weaker. This acid washing is made ammoniacal and poured on paper with the main precipitate. The main part of the blue color should now be washed out. If necessary, a little hot ammoniacal solution, 1 part of ammonium hydroxide to 1 part of water, poured around the edge of the paper, will greatly hasten this operation.

This ferric hydroxide precipitate, which contains all of the arsenic as ferric arsenate and all the antimony, probably as an ammonium antimoniate occluded by the voluminous iron precipitate, is dissolved

through paper with 3 cc. of nitric acid and 5 cc. of sulphuric acid which has been diluted to nearly 50 cc. with water and heated to boiling. Wash well with hot water. It is well to reserve a little of the hot acid solution to pour around the edges of the paper after washing with water. If any yellow color should show in the paper, wash again with hot water.

All of the copper that has been held by the ferric hydroxide must now be removed by electrolytic deposition on a small platinum gauze cathode having the following dimensions: height, 2.25 in.; diameter, 1.5 in. This cathode, as employed in our laboratory, is constructed of 52 mesh gauze and has supporting rings of heavy wire at top and bottom. The gauze is made of wire 0.0040 in. in diameter.

The solution, containing the iron, arsenic, antimony, and a small amount of copper, is heated quite or almost to boiling and a current of 0.5 ampere is passed for 2 hr. After about 1.5 hr., the sides of the beaker are washed down with a jet of water. This is to redissolve and remove any copper that may have spattered up from the main solution. At the end of 2 hr., the beaker is removed as quickly as possible, in order to avoid any re-solution of copper. It is unnecessary to wash the anode and cathode, as the small amount of solution adhering thereto will have absolutely no effect on the result, while attempting to save this solution may ruin the determination by copper being dissolved from the cathode.

Now heat the solution, from which all the copper has been removed, to boiling to be sure of complete oxidation, render ammoniacal, boil to aid in settling of precipitate, and when well settled filter through a firm 11-cm. paper. Wash once or twice with hot water to remove nitrates. In washing iron precipitate both in this and in the original filtration, it is well to disturb the precipitate on paper as little as possible.

Dissolve the precipitate through paper with a jet of dilute hydrochloric acid, washing alternately with hot water and dilute acid until no yellow color shows in the paper when it is washed with acid. The dilute acid should be made up of about 1 part of concentrated acid to 4 parts of water. Add ammonia carefully and finally drop by drop until the solution is but very slightly acid. For best results in this neutralization and subsequent reduction the solution should be near boiling temperature.

The reduction may be effected by ammonium bisulphite or, as suggested in Lunge's *Technical Methods of Analysis*, Vol. II., Part I., sodium hypophosphite may be used. In either case no more than is necessary to effect reduction of the iron should be employed. When

ammonium bisulphite is used long boiling is necessary to remove excess sulphur dioxide, hence the necessity of extreme caution to avoid any excess of the bisulphite. However, if the hypophosphite be used no boiling is required and the solution may be transferred at once to the hydrogen sulphide supply. In either case, cool and pass hydrogen sulphide 15 or 20 min. Weigh sulphides the following morning. The sulphides are weighed on an asbestos pad in a platinum Gooch crucible. The pad is prepared of moderate thickness of the very best quality of asbestos. It is washed with alcohol, dried at 105° C., and after cooling in a dessicator for 15 min. the crucible containing the pad is carefully weighed. The sulphides of arsenic and antimony are filtered on to this pad and are very thoroughly washed with water to remove all ferrous and ammonium salts. Wash well with alcohol to remove all water and then with carbon disulphide. After washing with carbon disulphide, using about 20 cc., follow with a thorough washing with alcohol. Dry 1 hr. at 105° C., cool 15 min. in dessicator, and weigh again.

The weight of the sulphides multiplied by 2.4 will give the result directly in percentage of arsenic and antimony, providing 25 g. was taken for assay. This factor has been arrived at after a great deal of experimenting.

Some Recent American Progress in the Assay of Copper-Bullion.

BY EDWARD KELLER, PERTH AMBOY, N. J.

(Butte Meeting, August, 1913.)

THE ASSAY FOR COPPER.

SOMEONE some time ago remarked that some chemists still insist on telling us how to determine copper by the electrolytic method. The writer must confess that he believes that everything is not known definitely as yet as to how the exact amount of copper is determined in such material as purest commercial electrolytic copper. Some of us are not yet convinced that the pure copper atom is at all times deposited from an acid solution and that no oxygen or hydrogen will, under certain conditions, accompany it. Difficulties with impurer material are frequent, but erroneous results are not always apparent, since these do not speak so readily for themselves as those in the first case. By this is meant that if a chemist finds 100 per cent. of copper in electrolytic copper it is pretty certain that he perceives the result to be wrong, while if he finds 0.1 per cent. of copper too much in a very impure material he is far more likely to be unconscious of his error. These, in a wide experience, are frequent happenings and the writer has always looked with interest to publications on this subject. In turn, he feels justified in giving to others a few of his own observations.

Up to a few years ago there were two methods of electrolytic copper assay in technical use, which had in common, that the amount of copper deposited on the cathode did not exceed 2 g., and they differed in that, for the one a large sample[1] (20 to 80 g.) was taken from the general sample by a splitting device, dissolved, and from the solution a small portion (to contain 1 or 2 g. of copper) measured out for the electrolytic deposition; in the other method the copper was determined, generally in 1-g. portions of the separated coarse and fine portions of the sample, all of which passed a 16 or 20-mesh screen, being accomplished by a 40-mesh screen,

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[1] W. C. Ferguson gave a thorough description of the details and precautions necessary in this method. *Journal Industrial and Engineering Chemistry*, vol. II, No. 5, p. 187 (May, 1910).

and the average assay was figured according to the weight-ratio of the parts. To use larger quantities (5 g.) for electro-deposition on such copper as Lake or commercial electrolytic had been recommended some years ago,[1] but this had never been attempted in a systematic way with crude coppers.

When in 1909, John T. Stoddard[2] published a paper in which he pointed out the feasibility of rapid electro-deposition of metals, with analytical accuracy, on stationary gauze cathodes and without special stirring device for the electrolyte, it occurred to the writer that this idea, in properly modified form, could be taken up in the laboratories of the copper smelteries and refineries for the purpose of more accurate copper determinations in the metallic materials; however, not by reducing the time usually required for deposition, but by multiplying the quantity of sample taken for the assay. It was reasoned that gauze cathodes, for much work, would be too flimsy and changeable in weight through abrasion and that perforated platinum sheet would answer the purpose better. For certain work, cathodes with larger perforations had been recommended and used before by G. A. Heberlein[3] of Great Falls, Mont. It developed after a few trials that with the new perforated cathodes[4] 5 g. of copper, from crude-copper solution, could be as quickly and cohesively deposited as 1 g. on the smooth cathode, with equal gross surface, within the customary time of 18 to 20 hr. With the perforated cathode a current of 0.5 ampere was applied, while with the smooth cathode 0.15 ampere had been found to be the upper limit for good work. It was found impossible to deposit, in good form, 5 g. of copper on a smooth cathode within 24 hr. Most important with the 5-g. method is the fact, that the general limit of agreement between duplicate determinations was moved from the second percentage decimal to the third, whatever the absolute accuracy might be. This in itself is sufficient proof of the fairness of each of the 5-g. charges taken from the general sample. Besides, it is the writer's opinion that there is far more simplicity in this method than in the one in which a much larger sample is dissolved and an aliquot portion of the solution, with a small quantity of copper (1 to 2 g.), measured out.

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[1] George L. Heath, The Electrolytic Assay as Applied to Refined Copper. *Trans.* XXVII, p. 390 (1897).

[2] Rapid Electro-Analysis with Stationary Electrodes. *Journal American Chemical Society*, vol. XXXI, p. 385 (1909).

[3] Frank Klepetko, Discussion of Heath's Paper on the Electrolytic Assay, etc. *Trans.* XXVII, 967 (1897).

[4] NOTE: As no priority controversy is desired, the writer would state that he placed his order for perforated cathodes with Baker & Co., Inc., Newark, N. J., on March 20, 1909, and that he exhibited them before a meeting of the New York Section of the American Chemical Society on March 11, 1910.

. The operation of this method for Anaconda copper, is now carried out in the following manner:

Stock solutions:—Nitric acid (sp. gr. 1.42) one part to one part of water; sulphuric acid (sp. gr. 1.84) one part to two and one-half parts of water; sodium chloride, 5.5 g. to the liter. The proper ratio of the fine portion of the sample, $\dfrac{5}{\dfrac{C}{F}+1}$ grams (C=weight of coarse; F=weight of fines in grams), is weighed and the remaining part of the 5 g. made up with the coarse portion. This is dropped into a 300-cc. lipless Jena beaker (tall form) which seems to be the smallest size that can safely be used for the solution of 5 g. of copper. Tests made by placing a filter paper over cover glass and beaker and snugly folded down over the edge of the latter
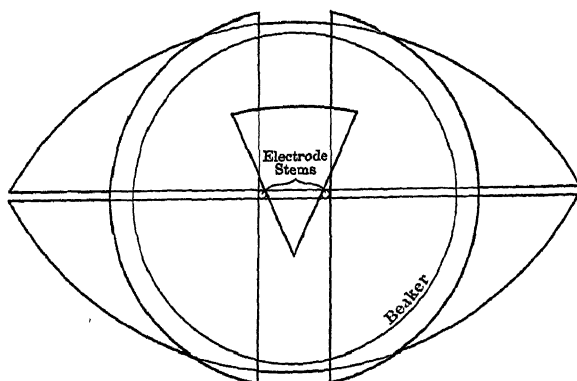
FIG. 1.—SYSTEM OF COVERING ELECTROLYTIC BEAKER.

revealed no loss of copper mechanically carried away. Ten cubic centimeters of stock sulphuric acid is first added to the copper, followed by 25 cc. of stock nitric acid; the purpose of the initial sulphuric acid is to retard the action of the nitric acid but afterwards to aid in the solution of the oxides present. The beaker, covered with a watch glass, is placed in a cool part of the hood until the copper is dissolved and is then placed on a steam plate, carefully avoiding boiling, until complete disappearance of nitrous fumes; 2 cc. of the sodium chloride solution is now added and the hot solution gently shaken until the silver chloride is well coagulated. Filtration is then performed through a 7-cm. filter into a beaker of the same size and shape as the original. To make up the full acidity of the electrolyte 30 cc. more of the stock sulphuric acid and 3 cc. of the nitric is necessary, all of which is run through the filter to insure complete recovery of the copper. The whole is diluted to 225 cc. with distilled water.

When any difficulty in depositing the copper is experienced it may be attributed to insufficient dilution of the electrolyte; addition of more water will overcome it.

The perforated platinum sheet constituting the closed cathode cylinder is 6 x 11 cm. and has seven perforations of about 0.5 mm. to the linear cm. The total height of the cylinder and stem is 23 cm.; they are somewhat unnecessarily heavy, weighing about 23 g. Those cylinders that have been in daily use for four years on Anaconda material lost on the average 1.7 milligrams per year. Others, which are used for copper high
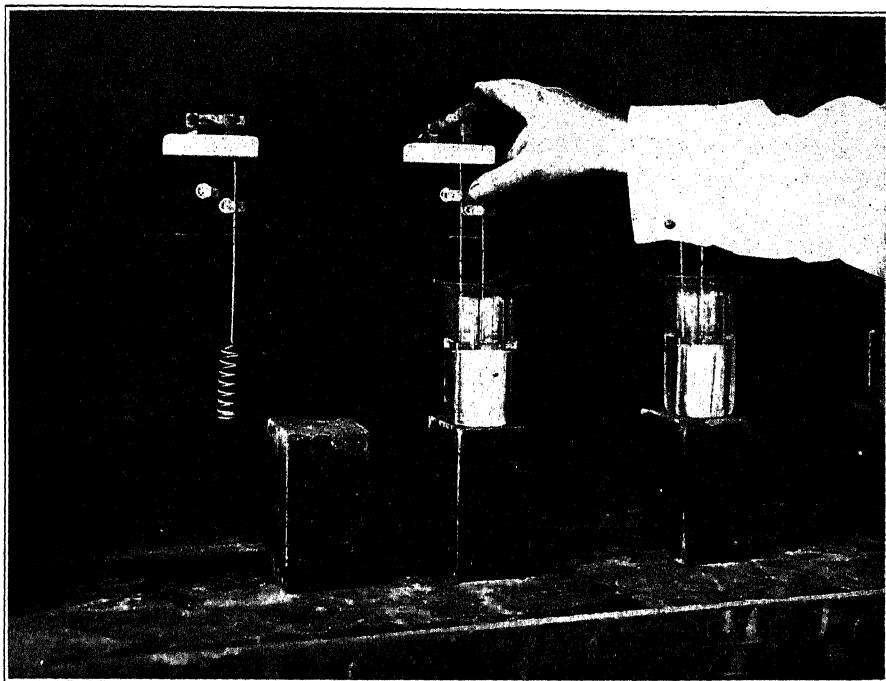
FIG. 2.—ELECTROLYTIC COPPER DEPOSITION STAND.

in gold, gain very appreciably in weight after every deposition; this being due to the deposition, with the copper, of that portion of the gold which is soluble in nitric acid. In course of time these cylinders become gold-plated. With the heavy current used, it is important that the beakers are well covered, so that there can be no loss of solution by spraying. Fig. 1 illustrates the scheme to accomplish this with the aid of several sets of split watch glasses. Fig. 2 shows the construction of an electrolytic stand with series connections. The ammeter is not visible. It is here shown chiefly to call attention to the knife switches on the top of the bar which

enable the operator to remove the cathode and simultaneously to close the circuit, which is not possible with the generally used plug. It is of the utmost importance to transfer the cathode with its copper deposit as quickly as possible from the electrolyte into pure water, so as to lose a minimum of copper by re-solution. It is done as follows: The covers are first removed from the beaker, then the base block from under the latter with the right hand, holding the beaker with the left. Up to this moment the current passes uninterruptedly. Now the forefinger of the right hand closes the switch and simultaneously the thumb presses the spring releasing the cathode; the left hand at the same time draws the beaker and cathode down and away from the anode (the usual corkscrew spiral). By this time the right hand is free to transfer the cathode to several clean waters and to absolute alcohol, after which it is dried over a Bunsen burner.

As already stated, the current employed for each assay is 0.5 ampere, the time (over night) for complete deposition about 18 hours. The latter point is arrived at when polarization (bubbles) at the cathode has taken place for about one-half hour. The beaker is then filled with distilled water, so that the electrolyte touches the cover-glasses and the current permitted to continue for another half-hour. Each electrolyte is tested for the presence of copper by saturation with hydrogen-sulphide gas.

The question of the accuracy of results should in all cases be subjected to analysis. It has already been hinted that after all the copper has been deposited on the cathode, there is a possibility of re-solution when removing it from the acid electrolyte. Tests made on numerous individual determinations showed that the electrolytes contained 0.0092 per cent. of the original copper. Table I shows, on the other hand, that

TABLE I.—*Impurities Contained in Copper-Bullion and Deposited with Electrolytic Copper.* (*Analysis of 100 Grams.*)

	Lead, Per Cent.	Bismuth, Per Cent.	Antimony, Per Cent.	Arsenic, Per Cent.	Tellurium and Selenium, Per Cent.
Impurities in converter copper....	0.0103	0 0040	0.0630	0.0211	0.0072
Impurities in electrolytic copper..............	trace	0.0037	0.0031	0.0009	0.0016
Portion of impurities deposited [a]	?	92 50 [b] 100.00 ?	4.92	4.26	22.22

[a] Gold in this case was not dissolved and silver is completely eliminated.

[b] Bismuth is probably all deposited, the difference between the two figures of analysis being well within the limits of experimental accuracy.

the copper which is deposited is not entirely pure, that in this case it contained 0.0093 per cent. of its weight of impurities deposited from the electrolyte. These two figures balance each other very closely and in commercial work usually no corrections are made.

In many laboratories it has been customary to precipitate the silver electrolytically with the copper on the cathode and then to subtract from the gross percentage that of the silver as determined by separate assay. This works well up to a certain silver content; 0.3 per cent. or 100 oz. per ton probably being the limit. When under this method the first difficulties with a good copper deposit are experienced, it is well to eliminate the silver from the electrolyte. On a number of occasions this was the only precaution necessary to bring about an excellent deposit of electrolytic copper.

When certain impurities in crude copper, or copper-bullion, exceed a certain quantity, it becomes impossible to electro-deposit from its solution a copper of sufficient purity for commercial analytical requirement. We have already seen that bismuth is all deposited with the copper, but this metal is rarely met in disturbing quantity in converter copper. When met in larger quantity in black coppers it must be eliminated before the electro-deposition of the copper; a method for which will be described further on.

Selenium is an element which is more readily deposited from solution than copper and we meet it in appreciable quantity in the converter coppers from our Southwest. When the copper on the cathode is dissolved in dilute nitric acid the selenium remains, at least in part, as a red coating, and some chemists weigh this and use the weight as a minus correction in their copper assay. This, no doubt, is but an approximation. The following is a very simple method to remove quantitatively the selenium from any quantity of copper, the silver being removed at the same time.

Selenium and copper cannot be completely separated by the alkaline sulphide method, as there is always some insoluble selenide formed with the heavy metal. Selenium is also precipitated with the copper when the latter is separated as sulphocyanate from a number of other elements. In fact, however, any method that would necessitate the precipitation of 5 g. of copper as a chemical compound for the purpose of its purification would be unsatisfactory. The precipitation of the selenium from the electrolyte by means of sulphur dioxide naturally suggested itself, but it was soon found that here also the two elements combined to form copper selenide.

Upon further investigation it was learned that in the presence of copper and silver, in sulphate solution, the selenium, when liberated by sulphur dioxide, had a strong preferential affinity for silver and if the latter was

present, in sufficient excess, the selenium would be precipitated quantitatively as silver selenide, Ag_2Se, without a trace of copper. The excess of silver is then precipitated as chloride and with the silver selenide is filtered and washed with dilute sulphuric acid and water. In case too much of an excess of salt solution has been used, it is best now to evaporate to sulphuric acid fumes, in order to drive off the chlorhydric acid and thus avoid its deleterious effect on the character of the copper deposit.

The presence of an appreciable quantity of tellurium will also interfere with the accuracy of the electrolytic copper determination and it must, accordingly, be eliminated before the electro-deposition of the copper. However, it is not often present in disturbing amount. Tellurium is not very readily precipitated from the copper sulphate solution by sulphur dioxide, but whatever portion of it is reduced combines with copper to form copper telluride. It is no rival of selenium in the latter's affinity for silver. A complete elimination method will be described with that of the bismuth.

Antimony and arsenic are not very readily electro-deposited from nitric-acid solution and only small percentages of the original quantities are found in the electrolytic copper; yet they become troublesome at times, because they are generally the elements found in largest quantity associated with copper. It has been claimed that certain compounds, "dopes,"[1] added to the electrolyte will entirely prevent the deposition of these elements. In tests made in the Anaconda laboratory with commercial electrolytic copper it was found with the addition of such dopes the results were very noticeably high.

To eliminate antimony and arsenic when they become disturbing factors, the writer prefers the double-deposition method, which has proved to be very satisfactory and in proof of which the data of Table II are illustrative. To execute this method the original deposit on the cathode is treated with the usual precautions and is then placed, together with the anode, in a regular depositing beaker, the complete electrolyte added and the whole covered as per scheme of Fig. 1. Only gentle heat is applied to insure the re-solution of the deposit and the solution is digested long enough to expel the reduction products of the nitric acid. When, in these cases, the copper deposit has a slight coating of arsenic or antimony, there is no re-solution of the copper during the time of removal of the cathode-cylinder from the electrolyte, since the former elements are first subject to attack.

Lead is one more element which is usually present in copper in small quantity, but which, when present even in larger quantity, may be readily eliminated in the course of the regular routine of preparing the electro-

[1] George A. Guess. The Electrolytic Assay of Lead and Copper. *Trans.* XXXVI. 605 (1905).

analysis. In the first place a major part of it may be separated as sulphate from sulphate solution and the small remaining part can be electro-deposited from nitric acid solution as peroxide on the anode; it being only necessary that the latter presents sufficient surface for the adhesion of the peroxide.

There is yet to be described a method by which bismuth and tellurium may be eliminated and which will, at the same time, include the elements for which individual methods have been given. For this the writer be-

TABLE II.—*Arsenic in Original Copper-Bullion and in Copper of First and Second Electrolytic Deposition.*

Arsenic in Original Bullion, 2.609 per cent.

	Average, Per Cent.	Arsenic in First Deposit, Per Cent.
First Electrolytic Deposits, Copper, Per Cent.: 94 632, 94.678, 94.682 94 652, 94.690, 94 692	94.671	0.0534; 2.05 of original
First Electrolytic Deposit, Copper Dissolved for Second Deposition, Per Cent.: 94.594, 94.626, 94.576 94.590, 94.652, 94.666 94.652, 94.594, 94.724 94.824, 94.770, 94 770	94.6698	.
		Arsenic in Second Deposit, Per Cent.
Second Electrolytic Deposit, Copper, Per Cent.: 94.580, 94 550, 94 526 94.528, 94.558, 94.544 94.596, 94.572, 94.594 94.600, 94.526, 94.554	94.5607	0 00146; 0.056 of original. 2.73 of first deposit.

lieves the simplest process to be the precipitation with ferric hydroxide in slightly ammoniacal solution with an addition of ammonium carbonate to precipitate the bismuth. To carry out this method the copper is dissolved and the silver eliminated as usual, 0.1 g. of ferrous sulphate having been added to the 5-g. copper charge. The solution is made slightly ammoniacal amd a little ammonium carbonate added; it should then be kept near the boiling point for about 15 minutes. The ferric hydroxide carries down with it all of the selenium, tellurium, antimony and arsenic, but unfortunately, and this is the drawback of the method, it also carries with it an appreciable quantity of copper, so that after filtering and washing the precipitate with ammoniacal water it must be re-dissolved, re-precipitated, filtered and washed, and the whole procedure repeated four or five times to insure the recovery of all of the

copper. In each re-precipitation the same conditions are to be observed as in the first one. All the filtrates are united and the regular quantity of sulphuric acid is added. Should the volume of solution be too great it must be evaporated to the desired mark.

The marked progress in rapid electro-deposition of metals in recent years, by means of high current density and the use of rotating electrodes or the rapid circulation of the electrolyte, is well known. These methods, undoubtedly, have their high merit for special purposes, where time is the chief factor. They do not seem to have appealed to the chemists of copper works where large numbers of accurate determinations are required and in which the work of the electric current during the night requires no attention whatever, while rotating electrodes, motors, and violently stirred electrolytes would be sources of much concern.

THE ASSAY FOR SILVER AND GOLD.

From time immemorial the methods of assaying any materials for silver and gold were in reality nothing but laboratory smelting methods. When in 1885, the writer arrived in Butte, Mont., he found that these metals, contained in copper bullion, were still determined by the all-fire assay, both in the same charge. There seems to be no record as to when and where the so-called combination method (dissolving the copper in nitric acid and precipitating the silver as chloride, etc.) was first introduced; but in the early nineties of the past century it became evident that for the sake of accuracy of the gold assay this metal must be determined by the all-fire method which, on the other hand, was quite unsatisfactory for silver. The combination method became the standard method for silver and the all-fire the standard for gold. From that time on probably most assayers were longing for a reliable single method for both metals, since the simplicity of the combination method, with its accurate results for silver, contrasted strongly with the unchemical, tedious and expensive all-fire method for gold.

To many, the comparative cost of the all-fire and sulphuric-acid assay-methods may be of interest. For the Anaconda laboratory, of Perth Amboy, it has been calculated with accuracy, that the materials per assay, exclusive of those used in the parting for gold and for the correction-assay (these being approximately the same for both methods), amount to 8.5 cents for the sulphuric-acid method and 75.5 cents for the all-fire method; a saving of 67 cents per assay. The saving in labor, too, is considerable, although its value in our case has not been established.

The first recorded attempt to modify the combination method in a way to obtain accurate gold results was made by L. D. Godshall,[1] who

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[1] Assay of Copper Materials for Silver and Gold. *Trans.* XXX., 529 (1900).

suggested to reprecipitate any gold that might have gone into solution[1] with the copper by conducting hydrogen-sulphide gas into the solution in sufficient quantity to precipitate all the silver with no copper or only a minimum quantity thereof.   A. R. Ledoux and Cabell Whitehead[2] pointed out the defects of such a method.   About the same time W. R. Van Liew[3] showed that accurate gold results could be obtained by modifying the combination method in the way that dilute nitric acid was added in several portions to the metallic copper and the reaction conducted under ice-cooling.   The time required for this method was a factor very much against its becoming popular.   A new idea was introduced by Thomas B. Swift,[4] who first amalgamated the surface of the copper by means of mercuric nitrate before dissolving the metal with nitric acid. In this method, the surface of each copper particle would remain coated with mercury during the whole process of solution and the mercury would absorb the gold and protect it from any solvent action.   The author gave no data regarding silver.

During the period in which the afore-described efforts for a simplified gold assay were made, attempts along the same line were also carried on in the Anaconda laboratory.   The idea was to find a reducing medium which would quickly precipitate the gold from the nitric acid solution without precipitating copper in any form.   Good results were obtained by boiling the copper nitrate solution (one assay ton of copper), which contained little nitric acid in excess and no nitrous products, with 25 cc. of formaldehyde, strength about 37 per cent.   A few comparative results by methods of that time are given in Table III.

A peculiar phenomenon was observed in trying cane sugar for the same purpose; instead of reducing and precipitating the gold which had been dissolved by the nitric acid, the greater portion of the total gold would be dissolved when the solution was boiled; Table IV gives the data.

It was at first thought that chlorine was present in the sugar and that it was responsible for the solution of the gold, but a test of the sugar solution as well as of the burnt residue revealed but an indistinct trace of that element, so that the following facts may satisfactorily explain the phenomenon.   When the copper nitrate solution is boiled with the addition of sugar the latter is quantitatively converted into oxalic acid and this forms, with the copper, an insoluble oxalate.   The nitric acid

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[1] Edward Keller, The Solubility in Nitric Acid of Gold Contained in Copper Bullion, *Trans.*   XLIII., 582 (1912).

[2] Discussion of L. D. Godshall's paper, *Trans.* XXX., 1121 (1900).

[3] Losses in the Determination of Gold and Silver in Copper Bullion, etc.   *Engineering and Mining Journal*, vol. LXIX., No. 16, p. 469 (April 21, 1900).

[4] The Assay of Copper Bullion, *Engineering and Mining Journal*, vol. LXXIV., No. 20, p. 650 (Nov. 15, 1902).

TABLE III.—*Comparative Gold Assays by Various Methods.*

Gold, milligrams or ounces per ton.

SAMPLE No.	All-Fire Method	Swift's Method Hg = 0 75g.	Combination Method	Combination Method, Formaldehyde = 25cc
1.	5 82	5.88	5 70	5 82
2	6 03	6.08	5 88	6 02
3.	5 78	5.80	5 58	5 78
4	5 78	5 81	5 60	5 79
5	5 66	5 68	5 51	5 64
6	5 56	5 60	5 48	5 55
7.	5 41	5 50	5 42	5 45
8	5 63	5 66	5 52	5 71
9	5 44	5.56	5 32	5 47
AVERAGE.	5 6789	5.7300	5.5567	5 6922

is reduced to free nitrous acid and this will, the oxalic acid being inert, dissolve the gold.

Experiments on these lines were discontinued when, early in 1908, it was learned that Frederic F. Hunt,[1] of New York, had devised a new combination method which yielded good results for silver and for gold. This method consisted in adding to 1 assay ton of copper 10 cc. of water and 100 cc. of sulphuric acid of 1.84 specific gravity; first heating on a hot-plate and finally on an open flame until complete sulphatizing of the copper was accomplished. There was no solution of the gold and when the silver, or a part of it, was dissolved it was precipitated as chloride and the remaining procedure was the same as that well known in the old combination method. Some difficulty was experienced in sulphatizing the copper to completeness when the sample was too coarse and when

TABLE IV.—*Gold-Solubility in Copper-Nitrate Solution in the Presence of Cane Sugar.*

(Gold present = 4.16 mg.)

Grams, sugar.	1	2	3	4	5
Gold after boiling, mg	0.69	0.31	0.30	0.39	0.33

the copper contained appreciable amounts of sulphur. A little later in the same year Albert M. Smoot,[2] of Ledoux & Co., New York, informed the writer that copper first amalgamated superficially and heated with

[1] Private communication; since published with modification, *Engineering and Mining Journal*, vol. LXXXVII., No. 9, p. 465 (Feb. 27, 1909).

[2] Private communication.

80 cc. of sulphuric acid (sp. gr. 1.84) per assay ton of copper was an improvement over Hunt's method. Difficulties in the complete sulphatizing of the copper in certain bullions continued and led to numerous tests in the Anaconda laboratory which resulted in the final adoption of the following routine, by which even coarse drillings and copper shot may be completely sulphatized and of which the resulting gold and silver chloride may be scorified with a total of 20 g. of lead in 2-in. scorifiers and cupelled in 1-in. cupels; the lead buttons weighing 5 g.

Stock solutions.—Mercuric nitrate, 25 g. of mercury per liter; sulphuric acid, sp. gr. 1.84; sodium chloride, 19 g. per liter (10 cc. will precipitate 350 mg. of silver). The copper bullion drilling sample has been ground to pass a 16-mesh screen and if by test it has been found that the coarse and fine parts, separated by a 40-mesh screen, differ appreciably in precious-metal content the parts are weighed separately in their proper ratio. The fine, $\dfrac{29.166}{\dfrac{C}{F}+1}$ grams (C = weight of coarse; F = weight of fine portion of sample), is weighed first and the remainder of the assay ton made up with the coarse. This is now placed in an 800-cc. Jena beaker, Griffin shape, 30 cc. of water and then 10 cc. of the mercuric-nitrate solution added (Hg = 0.25 g.). The beaker is shaken until all the copper appears amalgamated over its surface, then 100 cc. of sulphuric acid is added; the beaker is covered with a 5-in. watch glass and placed on an electric hot-plate. The time necessary to complete the reaction depends on the state of division of the sample, as also on the temperature of the plate. Table V. gives the time and temperature record of two samples.

For about one hour the liquid appears to boil, which, however, is only a bubbling, due to the evolution of sulphur-dioxide gas, from the reduction of the sulphuric acid and the oxidation of the copper. This completed, the supernatant liquid assumes a very dark green color, finally changing to a light grayish blue which, as experience has taught, is the indication of the finishing point. Boiling over an open flame has been abandoned, the excessive heat having been found unnecessary and the act of boiling being bound to entail some loss. The beaker is next removed from the plate and placed to cool on an asbestos sheet. Complete cooling is unnecessary as there is very little free sulphuric acid present. Four hundred and fifty cubic centimeters of water with the necessary amount of sodium chloride solution, the latter depending on both the amount of the silver and the amount of mercury present, are used. With 80 to 100 mg. of silver and 0.25 g. of mercury for Anaconda material 30 cc. of the stock solution or 0.57 g. of sodium chloride is always a safe quantity. The beaker is again placed on the hot-plate and the solution brought to a

TABLE V.—*Time and Temperature-Record of Sulphatizing Amalgamated Copper-Bullion Samples (One Assay Ton).*

SAMPLE I			SAMPLE II		
Time, p. m.	Temperature, F.	C.	Time, p. m.	Temperature, F.	C.
Start....... .. 2 35	302	150	Start...... .1 50	224	107
2.38	406	208	1 57	336	169
2 45	412	211	2 05	350	177
3 05	412	211	2.15	358	181
3 25	378	192	2 25	364	184
3 45	386	207	2.35	378	192
3 55	384	206	2 45	374 .	190
4.05	386	207	2.55	389	198
4 15	408	209	3 05	390	199
4.25	415	213	3 15	386	197
Finish........4.35			3 25	389	198
			Finish...... 3 27		
Total time 2.00			1 37		

boil, which dissolves the copper sulphate and coagulates the silver chloride. On removal of the beaker 150 cc. more water is added, the total amount now being 600 cc., which is capable of keeping in solution all copper sulphate after cooling. It is then a matter of expediency to filter immediately or the day after.

Details as to the furnace operations of this method are not deemed to be within the scope of this paper; they are much a matter of ·individual taste. For the description of mechanical and labor-saving devices the reader is referred elsewhere.[1]

It should be noted that with comparatively pure copper the amount of mercuric nitrate may be reduced, while with copper high in sulphur content an increase in the amount of the mercuric nitrate will be required.

It has been noted as a peculiarity of this method that with certain low-grade bullions none of the silver is rendered soluble and that upon addition of sodium chloride solution no silver chloride is formed. Table VI. gives a variety of data on this subject. It is shown that all of the arsenic and nearly all of the antimony is dissolved. It was also qualitatively proven that no selenium or tellurium remains in the residue. The insoluble silver, therefore, is not combined with any of these elements.

It was found that with all of the insoluble silver residues there remained some mercury and with sample No. 4 always a little copper; the latter probably being in the form of sulphide, as this sample contained

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[1] Edward Keller, Labor-Saving Devices in the Works Laboratory, *Trans.*, XXXVI., 3 (1905); XLI., 786 (1910).

TABLE VI.—*Soluble and Insoluble Silver by Sulphuric-Acid Assay-Method: with and without Mercury.*

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver,[a] oz. per ton	579.33	131.30	710.63	84.33	129 83	214.16	53 66	35 95	89 61	6 90	28 39	35 29
Arsenic, per cent	2.506	0.00	2.506	0.0444	0.00	0.0444	0 0589	0 00	0 0569	0 1558	0 0	0 1558
Antimony, per cent	0.0885	0.0060	0.0945	0 1600	0 0054	0.1654	0 0355	0 0060	0 0415	0 0344	0 0062	0 0406

Mercuric Nitrate, 20 cc.; H₂SO₄, 100 cc.

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver, oz. per ton	477 14	236.01	713 15	164 62	48 47	213 09	71 69	17 46	89 15	23 83	10 85	34 68

Mercuric Nitrate, 10 cc., H₂SO₄ 100 cc.; heated 18 hr.; 15 mn. over open flame

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver, oz. per ton	621.50	89 35	710 85	......	......	.	86 19	3 88	90 07	23 30	11 60	34 90

No Mercuric Nitrate; H₂SO₄ 100 cc.; heated 18 hr.; 15 min. over open flame

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver, oz. per ton	475 04	235.97	711.01	209.28	6.44	215 72	1 22	88 75	89 97	0 38	35 18	35 55

Mercuric Nitrate, 10 cc.; H₂SO₄ 200 cc.; heated 18 hr.; 1 hr. over open flame

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver, oz. per ton	708.82	8.86	712 68	b209.30	6 36	215.66	88 62	1.42	90.04	b37 60	0 48	38 08

No Mercuric Nitrate; H₂SO₄ 200 cc.; heated 18 hr.; 1 hr. over open flame

SAMPLE	1 Soluble	1 Insoluble	1 Total	2 Soluble	2 Insoluble	2 Total	3 Soluble	3 Insoluble	3 Total	4 Soluble	4 Insoluble	4 Total
Silver, oz. per ton	285.66	425 98	711 64	5.98	211.12	217.10	1.92	88.38	90.30	0 18	37 90	38 08

[a] Assayed by our regular sulphuric acid method; 30 cc. water, 10 cc. mercuric nitrate solution, 100 cc. H₂SO₄ (sp. gr. 1.84). The volume of water used in all of the above assays was constant, 30 cc.
[b] Nos. 2 and 4 samples changed.

TABLE VII.—*Gold and Silver in Copper-Bullion by the Sulphuric-Acid Method. With Varying Amounts of Mercury for the Amalgamation of the Copper.*

Weight, Milligrams or Ounces per Ton.

Volume of mercuric-nitrate solution and water constant (35 cc.).

	None	0.125	0 250	0 500	1	2	4
Mercury, grams	None	0.125	0 250	0 500	1	2	4
Water, cc.	35	30	25	15	25	25	25
H₂SO₄ (1 84), cc	100	100	100	100	100	100	100
^{a}Salt solution, cc	12	12	12	15	40	80	120
	66.68	67.06	67 20	66.66	67.16	67.22	66 96
	66.96	67.16	67 20	66.54	67.32	67.18	66 82
	67.16	67.22	66 92	66.48	67.16	67.22	66 92
	67.32	67.02	66.96	66.72	67.42	67.02	66 90
	66.88	67.04	67 34	66.76	67 14	67.00	66 92
	66.50	66.78	66 98	66.76	67.24	67.06	66 92
	66 64	66.94	67.26	67.24	67.18	67 20	67 10
	66.90	66.92	67 12	66.83	67 64	66 98	67.14
	66 72	67.06	66 96	66 72	67.32	67.36	67.12
	66.50	67 00	67.16	66 74	67.02	67.20	66 80
	66 70	67 32	67.26	66.10	67 36	67 14	
	66 98	67.02	67.18	66 58	67.22	67 04	
	67.14	66 84	67.28	66 66	67.16	67 20	
	66 84	66 90	67 14	66.76	67 42	67.34	
	66.76	66 68	67 18	66 84	67.50	67 30	
	67.06	66 96	67.56	66 78	67 30	67.34	
	67.20	66.86	67.42	66.70	67.42	67 14	
	66.94	66.76	67.18	66.58	67.36	67.18	
	67 26	67.26	67.04	66.64	67.38	67 34	
	67.38	66.68	66.68	66.72	67.36	66.82	
Average, silver-gold	66 926	66.974	67.151	66.690	67.304	67.164	66.960
Less gold	0.476	0.476	0 479	0 479	0.483	0.476	0 483
Silver	66.450	66.498	66.672	66.211	66.821	66.688	66.477
Silver correction	0.906	1.128	0.999	1.222	1.009	1.069	1.130
Corrected silver	67.356	67.626	67.671	67.433	67.830	67.757	67.607

GOLD ASSAYS.

Uncorrected	0.4760	0.4765	0.4785	0.4795	0.4830	0.4755	0.4830
Correction	0.0010	0.0015	0.0020	0.0015	0.0010	0.0010	0.0010
Corrected gold	0.4770	0.4780	0.4805	0.4810	0.4840	0.4765	0.4840

^{a} One cubic centimeter of salt solution contained 0.019 g. sodium chloride.

by far the most of that element. We may conclude that, although the presence of mercury facilitates the solution of the silver as it does that of the copper, all copper and all mercury must be dissolved before all silver can go into solution. In the ordinary process of the assay, there is not enough sulphuric acid present and the heating is not continued sufficiently long to meet these conditions; therefore, with little silver present none is dissolved, and with much silver present only a portion is rendered soluble.

Table VII. demonstrates that an increased amount of mercury has no marked effect on the silver results if only the necessary increase in the amount of sodium chloride be provided. Neither of the two substances can influence the gold results.

Table VIII. demonstrates that immediate filtration of the silver chloride may be chosen for ordinary purposes, while when great accuracy is desired it is safer to leave it for settling and complete cooling over night.

TABLE VIII.—*Experiment on Sulphuric-Acid Method Silver Determinations. Filtration Test.*

Milligrams or ounces per ton.

Sample No.	Filtered Next Day	Filtered At Once	Difference		Sample No.	Filtered Next Day	Filtered At Once	Difference	
			Over	Under				Over	Under
1	84.92	84 89	0.03	.....	19	87.13	87.18	.....	0 05
2	81 87	81.79	0.08	.....	20	85.44	85.41	0.03	.....
3	80.28	79.97	0.31	.....	21	84 39	84.66	....	0.27
4	84.46	84.13	0.33	.....	22	87.82	87.63	0.19	....
5	82.91	82.73	0.18	.....	23	85.35	85.00	0.35	.....
6	82.86	82.60	0.26	....	24	90.01	90.07	....	0.06
7	90.01	90.06	....	0.05	25	87.39	87.06	0.33	.....
8	83 97	83.78	0.19	.....	26	86.72	86.43	0.29	.....
9	82.46	82.40	0.06	.....	27	90.81	90.59	0.22	.....
10	90.62	90.72	.....	0.10	28	85.62	85.25	0.37	.....
11	89.10	88.63	0.47	.....	29	84.04	84.51	....	0.47
12	87.61	87.50	0.11	.....	30	86.14	85.83	0.31	.....
13	87.42	87.56	.....	0.14	31	89.15	89.39	.....	0.24
14	84.76	84.50	0.26	.....	32	85.86	85.59	0 27	.....
15	84.12	84.30	.....	0.18	33	86.28	85.91	0.37	.....
16	86.40	85.83	0.57	.....	34	87.22	86.77	0.45	.....
17	85.14	85.09	0.05	.....	35	86.51	86.34	0 17	.....
18	86.95	86.54	0.41	.....					

Average............. 86.0497   85.9040   0.1903   0 0446
Difference.......... 0.1457   0.1457

Table IX. gives comparative results for silver and gold; the former by two and the latter by four methods.

In Table X. are given probably somewhat more convincing, comparative figures for gold as determined by the once standard all-fire method

TABLE IX.—*Comparison of Several Methods for Assaying Copper-Bullion.*

Milligrams or ounces per ton.

	SILVER		GOLD				
	COMBINATION		COMBINATION				
Sample No.	Nitric-Acid Method	Sulphuric-Acid Method	Nitric-Acid Method		Sulphuric-Acid Method		All-Fire Method 0 1 Assay Ton
	Water	Mercury	Water	Mercury	Water	Mercury	
1	78 3	78 4	0 22	0 26	0 26	0 28	0 27
2	71 8	71 9	0 27	0 30	0.30	0.31	0 30
3	73.7	73.7	0 28	0.31	0 30	0 30	0 31
4	72 6	72.9	0 28	0.31	0 31	0.32	0.33
5	71 9	71 8	0.28	0.30	0 29	0 30	0 31
6	72 7	72.9	0 27	0.30	0 30	0 30	0 31
7	80.8	80 7	0.30	0.34	0.34	0 33	0 35
8	73 7	73.7	0.28	0.30	0.31	0.32	0.31
9	81.7	81.7	0 31	0 32	0.34	0.34	0 35
10	85 1	85 3	0.27	0 30	0.30	0.30	0 29
11	78.6	79 0	0 22	0 26	0.27	0 27	0.27
Average	76 445	76 545	0.2709	0.3000	0.3018	0.3064	0.3091
Average weighing all gold buttons together....			0 2655	0.2955	0.2955	0 2991	0.2973

TABLE X.—*Gold Assay Test.*

Weight, milligrams.

NUMBER OF ASSAYS		METHOD	
		Sulphuric Acid	All-Fire
	10	3.911	3.881
	10	4.162	4 122
	20	7.195	7.155
	29	10.384	10.344
	53	23 697	23.646
Total	122	49.349	49.148
		Ounces Per Ton	Ounces Per Ton
Average uncorrected..		0 40450	0.40285
Correction (slag and cupel losses).		0.00180	0.00380
Average corrected...........		0.40630	0.40665

and the now adopted sulphuric acid combination method. Clearly, the figures for the all-fire method are lower than those for the sulphuric acid method when the scorification and cupellation losses are not taken into account; but when corrections are made for these losses they may justly be deemed identical for all practical and commercial purposes.

This leads to the consideration of the corrected versus the uncorrected silver and gold assays. Nearly 20 years ago Carl Stetefeldt published a paper[1] in which he pointed out that, in assays for silver in silver and silver-lead ores the losses entailed in the commercial assay methods amounted to from 5 to 20 per cent., depending on the grade and composition of the ore, of which no account was taken. There can be little doubt that the silver losses in assaying copper ores are equal, if not greater, than those in Stetefeldt's ores. Unless such losses are carefully deter-

TABLE XI.—*Silver Losses in Scorification and Cupellation. Silver-Chloride Precipitated from Solution of Copper-Bullion.*

Mg. or Oz. of Silver per Ton found after Scorification and Cupellation	Mg. or Oz. of Silver per Ton found in Slag and Cupels (Correction)	Percentage of Silver Retained in Slag and Cupels
15.71	0.61	4 02
29.00	0.82	2.83
35.74	0 81	2.27
55.57	1.41	2.54
86.18	1.72	2.00
91.15	1.62	1.78
163.02	2 62	1.61
193.91	2.98	1.54
220.17	2 90	1.32
317.64	4.52	1.42
447.34	6.11	1 37
596.73	7.68	1.12
725.48	8.38	1.16
755.72	8.77	1.16

mined and the proper corrections made on all materials from the mine to the refinery, the statistics of milling, smelting and refining-efficiency must be a deception; they are unable to tell us the truth on conservation. In order to understand this it must be remembered that percentage-losses in assaying are by far the lowest in the high-grade or final products of the metallurgical establishments. For an example, let us take the loss for silver in these materials at 2 per cent. and that in the ore as probably not less than 10 per cent.; accordingly, the mill receiving the ore would be charged with only 90 per cent. of the silver which it has actually received. Suppose that in its highest product it shows, by the same assay

[1] The Inaccuracy of the Commercial Assay for Silver and of Metallurgical Statistics in Silver Mills, etc., *Trans.* XXIV, 530 (1894).

methods, a yield of 95 per cent., the apparent loss through its whole process would then be 5 per cent., while the true loss would be made up of the factors 10+5—2, or 13 per cent.

There are many data given in chemical and metallurgical literature on the assay losses of silver and gold, but Table XI., giving data on silver only, taken from actual practice in our own laboratory and derived from uniform methods, gives a wider range than anything the writer can now recall.

It clearly shows the slow increase in percentage loss toward the high-silver end, or the rapid increase of these losses toward the low-silver end of the table. Fig. 3 shows the same data plotted and the general trend illustrated by a curve. There are no corresponding data for gold available, but it may be stated that the loss percentages for that metal, in the materials and by the methods under discussion, are far smaller than for silver. A similar relation holds true as regards value losses of the two metals, which is illustrated in Table XII, the figures being taken from a full month's run of Anaconda copper-bullion. While these values look small when considered for the single ounce, they are of commercial importance when the great quantities produced per year are taken.

TABLE XII.—*Scorification and Cupellation Losses, Weight Percentage and Value Percentage for One Month's Run of Anaconda Copper Bullion.*
Value per ounce, silver = $0.60; gold = $20.67

	Average of Month Corrected Assay, Oz per Ton	Weight Correction, Oz. per Ton	Correction, Per Cent.	Correction Value per Oz., Cents
Silver	92.32	1.761	1.917	1.15
Gold.............	0.522	0 0016	0.031	0.64

Corrected assays should not only be made for commercial reasons on high-grade products, but they should be universally advocated for the sake of accuracy itself, for the establishment of correct efficiency records and true conservation statistics.

In making these assay-loss corrections for such uniform material as the Anaconda copper or that of other large producers it is perfectly permissible to collect assay slags and cupels for a whole month's run and to apply the correction found to the individual results of the month following. The mode of operation is as follows: The slag and cupels are crushed in a small jaw-crusher and thoroughly mixed. The whole or an aliquot part may be taken for the reduction-fusion in G crucibles, each of which is charged, in grams, with 200 slag and cupels, 70 borax, 70

bicarbonate of sodium and 10 flour. The resulting lead buttons are scorified and cupeled; the resulting silver-gold buttons by weighing and parting yield the corrections sought.

In order to obtain full recovery of the silver and gold from slags and cupels it is necessary to reduce practically all the lead from each crucible charge. For this 10 g. of flour with a salt cover suffice; the salt cover serves no other purpose than the prevention of combustion of the reducer.

Inquiry into the absolute accuracy of the heretofore-described assay methods naturally suggests itself. We have already seen the slag and cupel losses for silver and gold and have noted that corrections are made for them. There are, however, other losses, such as by volatilization, by solution of the silver chloride, and in mechanical ways; their values are
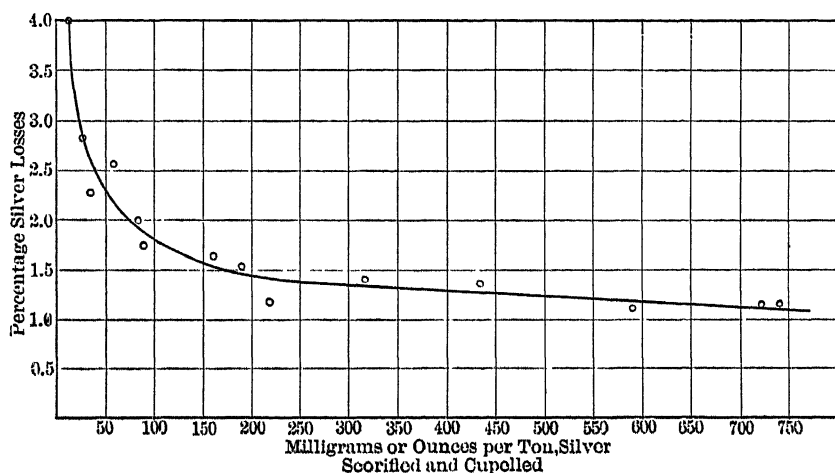
FIG. 3.—SILVER LOSSES IN SCORIFICATION AND CUPELLATION. SILVER CHLORIDE PRECIPITATED FROM SOLUTION OF COPPER BULLION.

of very difficult determination by direct methods. These losses are entirely or in part offset by the amount of impurities which remain with the buttons and are weighed as precious metals. It is a simple matter to analyze a sufficient number of silver buttons. Thirty grams of these, each weighing between 80 and 100 mg., were found to contain 0.16 per cent. lead and 0.15 per cent. bismuth, total 0.31 per cent., or the buttons to be of a fineness of 996.9. Table XIII. shows a test in which fine silver was subjected to exactly the same process as the silver in the assay of copper bullion, i. e., the fine silver was dissolved in a copper solution free from silver and precipitated as chloride, etc. It will be seen that the final and corrected result shows a small deficit, amounting to 0.098 oz. per ton, or to 0.12 per cent. of the total silver. More extended tests were undertaken with more silver and in comparison with higher grade materials,

anode residues or slimes. Here, too, the fine silver was subjected to the same treatment as the slimes, a sulphuric acid combination assay, and through the furnace process slimes and silver alternated in position, in scorifiers as well as in cupels. One-tenth of an assay ton of slimes was taken; all of the same sample. The fineness of the buttons was determined by titration by Volhard's method, with potassium sulphocyanate. The results of such a test are given in Tables XIV. and XV.

In these tables it will be observed that the large silver buttons are of slightly lower fineness than that which was given for those of lighter weight. Of chief interest is the fact, that when to the fine silver of the assay buttons of the slimes is added the slag and cupel correction and the volatilization and miscellaneous loss correction, as determined by the assay treatment of fine silver, the results correspond very closely to the usual results when the slag and cupel correction is added to the assay button of the slimes. Repeated tests confirmed this and the conclusion follows, that impurities in assay buttons and volatilization and miscellaneous losses balance each other. It follows that the assay of anode residues for silver by this assay method is correct in every case to within about plus or minus 0.1 per cent. of the total silver, an accuracy which could hardly be attained by any other chemical method. Of course, this degree of accuracy is only obtained with a sufficient number of determinations, averaged to one result, in our case 10, with 0.1 assay ton each of the material.

The contract assay methods for such materials are still of an archaic kind, of the all-fire nature, elastic beyond reason, in which results differing several per cent. are often obtained. It is to be hoped that concise methods will soon take the place of the old.

For the western (smeltery) and the eastern (refinery) end of the Anaconda Company, sampling has been established on a scientific basis and the most accurate assay methods are employed to determine the values of the copper-bullion shipped from one to the other. Any inaccuracy in sampling will offset the accuracy of analysis, or *vice versa;* an equal degree of accuracy is requisite for both. The concordance of results obtained in monthly averages at the two places is illustrated in Table XVI.

The data given in this paper were, in the greater part, determined and accumulated in the interest of the Anaconda Copper Mining Co., in its laboratory at the Raritan Copper Works, Perth Amboy, N. J. In this work the laboratory staff must naturally have contributed its share and the writer herewith especially acknowledges the services of K. W. McComas and W. L. Raup, Jr.

TABLE XIII.—*Experiment to Show Losses in Silver Assaying by the Combination Method.*

Fine Silver Weighed Out Milligrams	Weight of Assay Buttons Milligrams	Silver Recovered, Corrected Milligrams
81 62	80 28	81 61
83 74	82 34	83.67
83.44	82 18	83 51
93 62	92 24	93 57
83.56	82 18	83 51
81 04	79 40	80.73
85.20	83 72	85.05
85.82	84 28	85 61
85.18	83 66	84 99
84.40	82.78	84 11
83 42	82 24	83.57
87.40	86 00	87.33

Average silver weighed out, mg 84.870
Average weight of assay buttons, mg. 83.442

   Difference............. .... 1.428

Average silver weighed out, mg...... .. .. 84.870
Average silver recovered, corrected, mg...................... .... 84 772

   Difference......... .... 0 098
Correction for slag and cupel losses = 1.327 mg. = 1.33 mg.

TABLE XIV.—*Fine Silver Treated by Sulphuric Acid Assay Method.*
Silver titration by Sulphocyanate Method.

No.	Fine Silver Weighed Out Mg.	Assay Buttons Mg	KCNS 1 cc = 4.4907 Ag. Cc.	Fine Silver Mg.	Impurities Mg.	Difference Orig. Fine and Button Fine Mg.
1	374.28	370.24	82.1	368.69	1.55	5.59
2	376.94	373.38	82.9	372.18	1.20	4.76
3	373.47	369.68	82.1	368.69	0.99	4.78
4	374.26	370.06	82.1	368.69	1.37	5.57
5	373.96	370.72	82.2	369.14	1.58	4.82
6	369.81	366.56	81.3	365.09	1.47	4 72
7	371.73	367.76	81.4	365.54	2.22	6.19
8	373.96	370.56	82.1	368.69	1.87	5.27
9	370.25	367.02	81.3	365.09	1.93	5.16
10	372.70	368.74	81.6	366.44	2.30	6.26
Average...373.136		369.472	....	367.824	1.648	5.312
Slag and cupel correction..........		4.012		4.012	4.47 per M	....
Corrected Assays		373.484	....	371.836		

        Silver recovered by corrected assay................   373.484
        Original fine silver................................   373.136
        Difference..0.348 = 0.0933 per cent

        Original fine silver..............................   373.136
        Total silver recovered............................   371.836

        Volatilization and other losses...................   1.300
        Fineness of assay buttons........................   995.53

TABLE XV.—*Silver Assay of Anode Residues by Sulphuric Acid Method.*

Silver titration by Sulphocyanate Method.

No.	Assay Button Mg.	KCNS 1 cc = 4 4907 Ag. Cc.	Fine Silver Mg	Impurities Mg.
1..	371 22	82.2	369 14	2 08
2.	371 46	82 5	370 48	0 98
3......	371.42	82.5	370.48	0 94
4......	369 94	82 1	368 69	1 25
5.	370.82	82 3	369 58	1.24
6............... . .	370 90	82 2	369.14	1.76
7...............	371.44	82 3	369 58	1.86
8............ .	370 70	82 1	368.69	2.01
9............ . .	371.74	82 4	370 03	1.71
10...............	371.48	82.2	369.14	2.34
Average.........	371.112	....	369.495	1.617
Correction.....	3.860	.. .	3.860	4.36 per M.
Volatilization, etc.	....	.. .	1.300	....
Total............ ...	374.972	....	374.655	....

Difference..   0.317 = 0.106 per cent.
Fineness of assay buttons, 995.64.

TABLE XVI.—*Assay-Results in Monthly Averages, East and West, Obtained on the same Copper-Bullion.*

(About 125 samples assayed per month).

MONTH	WEST			EAST		
	Copper, Per Cent.	Silver, Oz. per Ton	Gold, Oz. per Ton	Copper, Per Cent.	Silver, Oz. per Ton	Gold, Oz. per Ton
1	99 3091	87.3093	0 4606	99.3139	87.2482	0.4592
2	99.3203	88.7305	0.6027	99 3310	88.7898	0.6057
3	99.2816	91.2517	0.4786	99.3091	91.1501	0.4817
4	99.2763	91.7445	0 4355	99.3073	91.5903	0.4382
5	99.2661	87.0271	0.4880	99.3061	87.0271	0.4877
6	99.2613	89.2657	0.5145	99.2979	89.1750	0.5143
Average[1] ..	99.2843	89.2620	0.4955	99.3101	89.2006	0.4966
West over. ...	.. . .	.. ..	....	....	0.0614	....
East over.....	0 0258	....	0.0011	....	....	....

[1] Correct average by weights.

DISCUSSION.

GEORGE L. HEATH, Hubbell, Mich.:—In the admirable paper quoted, is mentioned the date when the 5-g. assay was first proposed as a standard for refined metal.

The author omitted, however, to mention a later paper,[1] which contains a proposal for the use of a single "stock solution" of mixed acids, which has been adopted by the American Brass Co. and others.

The suggestion is made that such a scheme would save time and manipulation with converter metal. Unless the low current of 0.5 ampere and the large excess of nitric acid, recommended by Mr. Keller, have been proved necessary to hold up antimony and selenium, the use of the mixed solution which we have adopted will permit the use of 1 ampere of current per square decimeter of cathode surface, until the solutions have become colorless. At this point, we wash the covers and reduce to 0.5 ampere to prevent oxidation or contamination. The ordinary split electrodes have been used. With strong current, at least, the use of an excess of nitric acid tends to the neutralization of the solution before the end of the period, from the formation of too much ammonia, and this is detrimental. Antimony deposits more easily than arsenic, and tends to drag the latter down with it. There must be some such explanation for the fact that we have made hundreds of deposits of pure cathodes from Lake material containing no antimony, but as high as 0.5 per cent. of arsenic. By placing an assay beaker in the Frary solenoid, or rotary device, a cathode deposit with less than 0.01 per cent. of impurity is easily obtained in one operation from 3 g. of whitneyite, carrying from 7 to 10 per cent. of arsenic. In such a case, it is necessary to test several times for end point and remove the plate as soon as no brown tint appears (at once) when hydrogen sulphide water is added to the test portion on the spot plate.

Edward Keller's proportions of solvent figure out as 14 cc. of strong nitric to 11.4 cc. of sulphuric acid (1.84).

Our stock solution is made up in the following proportions by volume: 7 cc. of nitric acid (1.42), 10 cc. of sulphuric acid (1.84), and 25 cc. of water. Ordinarily, 40 cc. is taken for a 5-g. sample, unless the arsenic is over 0.5 per cent., in which case the amount of solvent is increased.

It may not be generally known that one chemist, Ferdinand Andrews, of the Raritan Works, is depositing regularly 10 g. of copper from refined metal in 16 hr. with the perforated cathode, described

[1] *Journal of Industrial and Engineering Chemistry*, vol. iii, No. 2, p. 74 (Feb., 1911).

by Mr. Keller, and a double anode, consisting of an inner spiral and outer frame in one piece, similar to the Hollard type advertised by Eimer & Amend, New York. The current strength is 1 ampere, and the first solvent is pure nitric acid, to which is added 6 cc. of sulphuric acid and 10 cc. of ammonia, after the nitrate has been evaporated to the crystallizing point. Here, as in the single-mixture method, ammonium salts are depended upon to assist in holding back the impurities.

If the same quantity of sample can be plated out with half the current required by the solid cathode, or if 10 g. can be taken, the perforated cathode certainly offers advantages in the assay of refined metal as well as crude bullion.

Removal of Selenium and Tellurium.—When these elements are the principal impurity, they have for years been successfully removed by a very simple scheme which takes care of Mr. Keller's objection that the metals, under certain conditions, form compounds with silver, or copper, when reduced with sulphur dioxide.

The nitrate solution is evaporated with excess of sulphuric acid until the dry residue is white. Dissolve in 60 cc. of water and wash into a lipped beaker, placing the tall beaker under a funnel fitted with a 3-cm. filter. Heat the solution to boiling, remove from plate, and charge with SO gas for 10 min. Gas, free from chlorine, is generated by slowly dropping a saturated solution of C. P. sodium sulphite from a separatory funnel into a large round-bottom flask half filled with concentrated C.P. sulphuric acid. After settling a few hours, at least, filter into the original electrolytic beaker, and wash free from acid with hot water. Boil gently to remove most of the sulphur dioxide gas. Ignite the filter in a porcelain crucible at a red heat until the residue is oxidized, to volatilize the harmful elements.

Re-dissolve the residue in 1.5 cc. of nitric acid and add to the main solution, and electrolyze in the usual manner. A little more nitric acid and 10 cc. of ammonia may be added, if necessary, to hold back antimony and bismuth.

Hydro-Electric Development in Montana.

BY MAX HEBGEN, BUTTE, MONT.

(Butte Meeting, August, 1913.)

CONTENTS.

IT is estimated that the total stationary power now used in the United States, steam, water and gas, is probably over 30,000,000 h. p. The total developed water power is about 6,000,000 h. p. The water power in the United States now economically capable of development probably exceeds 25,000,000 h. p. The total developed and undeveloped power is
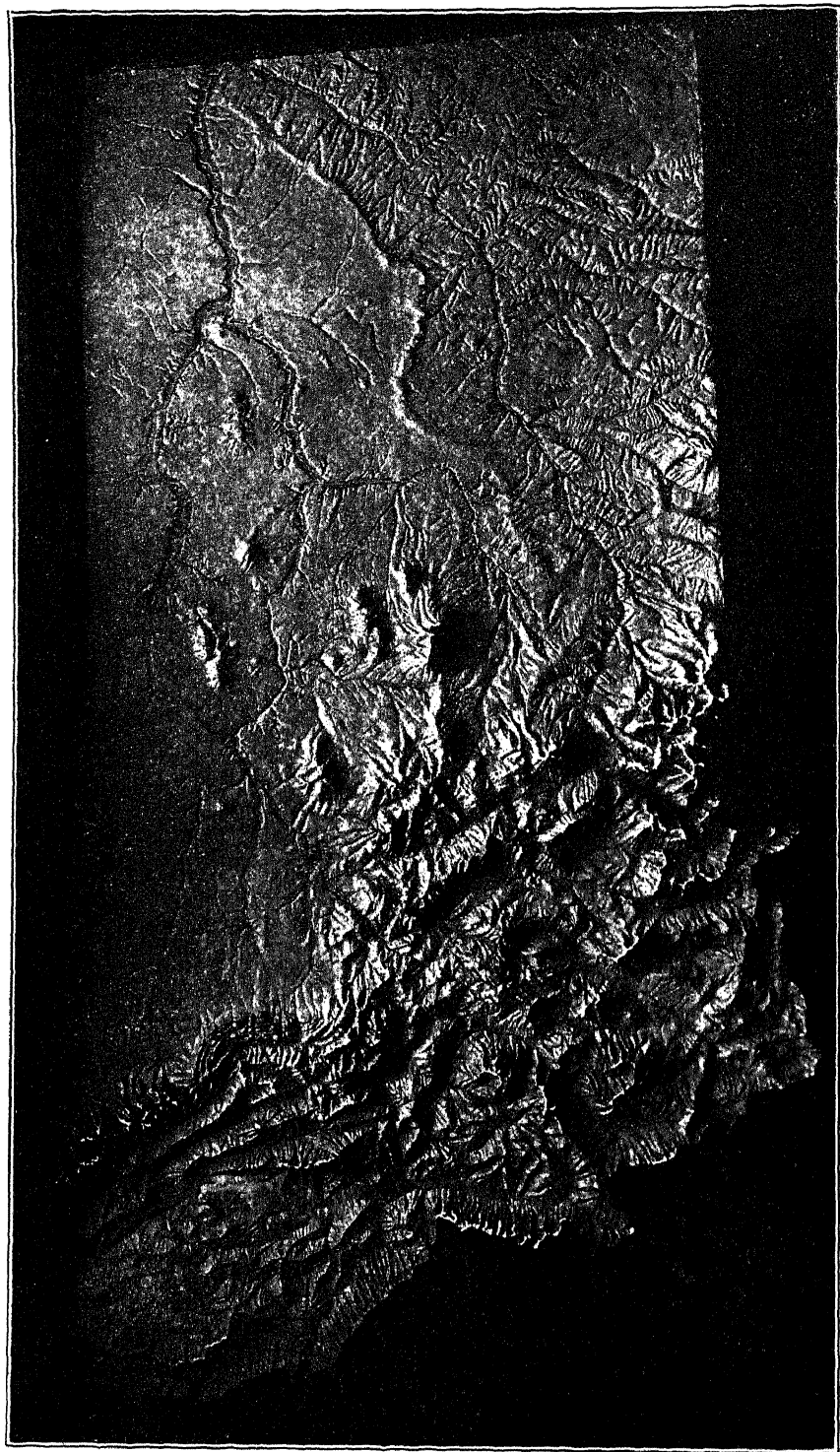
Fig. 1.—Relief Map of Montana.
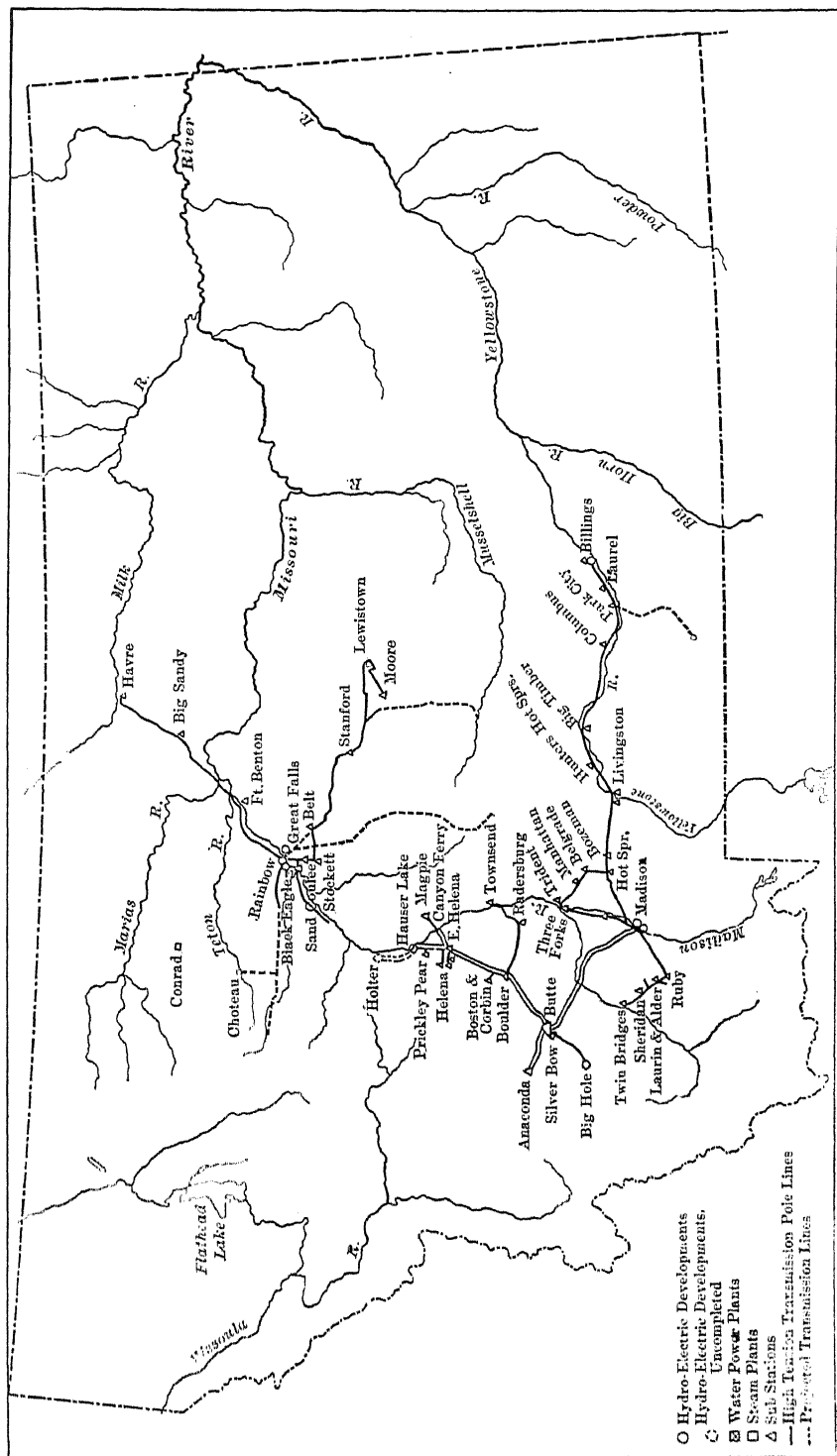
Fig. 2.—Power Developments and Properties of the Montana Power Co., Great Falls Power Co., and Subsidiary Companies.

mainly in the far Western States, Lake States and certain Atlantic Seaboard States.

· Within the State of Montana the streams rise in the high mountains at an elevation of from 5,000 to 8,000 ft. These streams leave the State line both east and west at elevations from 3,500 to 2,400 ft. It is, therefore, apparent that all these streams have an average fall of 3,000 ft. within the confines of the State, and many opportunities are presented on all streams for the development of power, by taking advantage of the natural fall.

It is probable that 1,000,000 h. p. can be developed in the State of Montana.

I. Natural Features of State Affecting Power Development.

The main range of the Rocky mountains, forming the Continental Divide, cuts the State of Montana into two parts. The eastern part is drained by the Missouri river and its tributaries, the Madison, Jefferson, Gallatin and Yellowstone rivers. The western part is drained by tributaries to the Columbia river, the Clarks fork and the Kootenai river. The Clarks fork in turn is formed by the junction of Missoula and Flathead rivers, the latter draining Flathead lake.

The western part of the State is mountainous and the eastern part is comparatively level. As a natural result the rivers in the western part are capable of generating large amounts of power, while in the eastern part of the State natural power sites are much fewer and farther apart. It also follows that in the mountainous districts large amounts of power are required for the mining industry, while in the eastern part of the State, which is principally devoted to agriculture, power is required in smaller quantities.

II. Early Developments.

1. Big Hole Plant.

One of the first power developments in the State was made on the Big Hole river about 22 miles south of Butte. This plant was built in 1899, and has a capacity of 3,000 kw. The development was made by building a rock-filled crib dam across the narrow canyon through which the river flows and developing a head of 65 ft. This plant has been in operation for 14 years and is apparently in as good condition to-day as the day it was built.

It supplies power to the city of Butte over a wooden-pole line operating at 15,000 volts. Power is used for lighting and for the operation of the street railway. Power is also supplied to the pumping plant of the Butte Water Co. situated on the Big Hole river about one mile below
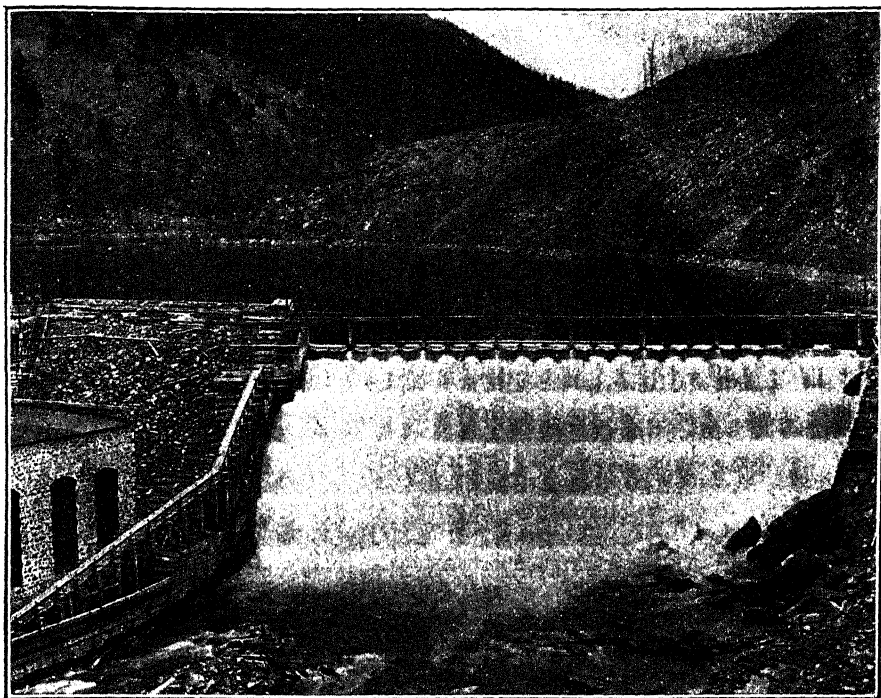
FIG. 3.—BIG HOLE DAM.
This dam is 65 ft. high and the plant has a capacity of 3,000 kw.

the power plant. A large part of the Butte city water supply is furnished by this plant through a wood-stave pipe line.

2. Canyon Ferry Plant.

This plant, situated on the Missouri river 17 miles from Helena, was built in 1898. A rock-filled timber crib dam, 39 ft. high, was built across the river and apparatus installed for 7,500 kw. Power was first transmitted to Helena at 11,000 volts, and a little later to Butte at 66,000 volts. Power from this plant is used principally to supply the requirements of the mining industry.

3. Madison Plant No. 1.

This plant was built in 1901. It is situated in the Madison River canyon, 61 miles east of Butte. This development originally consisted of a low crib dam and wooden flume about 1,000 ft. long supplying power to two 1,000-kw. units. The power was transmitted to Butte at 40,000 volts.
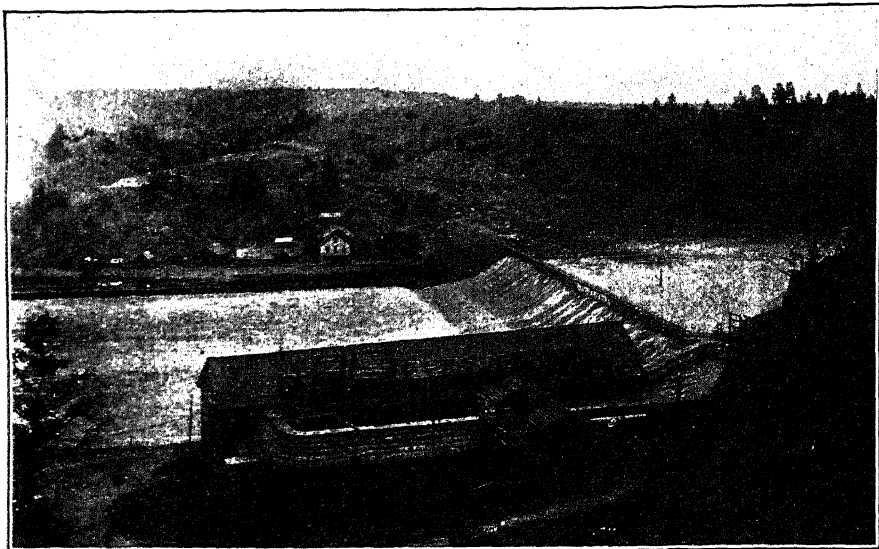
FIG. 4.—CANYON FERRY PLANT, 7,500 KW. CAPACITY.
Operates at 60,000 volts.
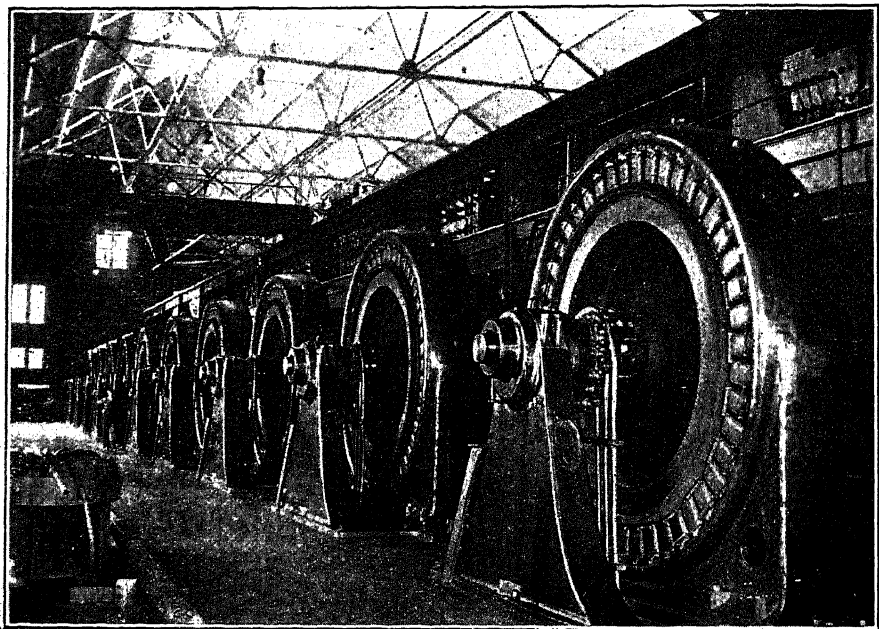
FIG. 5.—GENERATOR ROOM IN CANYON FERRY PLANT.
Ten 750-kw. generating units operating at 45 ft. head.

4. Black Eagle Plant.

In 1890 a dam was built across the Missouri river just above the Black Eagle falls, near the city of Great Falls, and power developed at this point to the extent of 8,300 h. p.; 7,500 of this was used directly by the Boston & Montana smelter and the rest was transmitted electrically and used for lighting and street-railway purposes in the city of Great Falls.
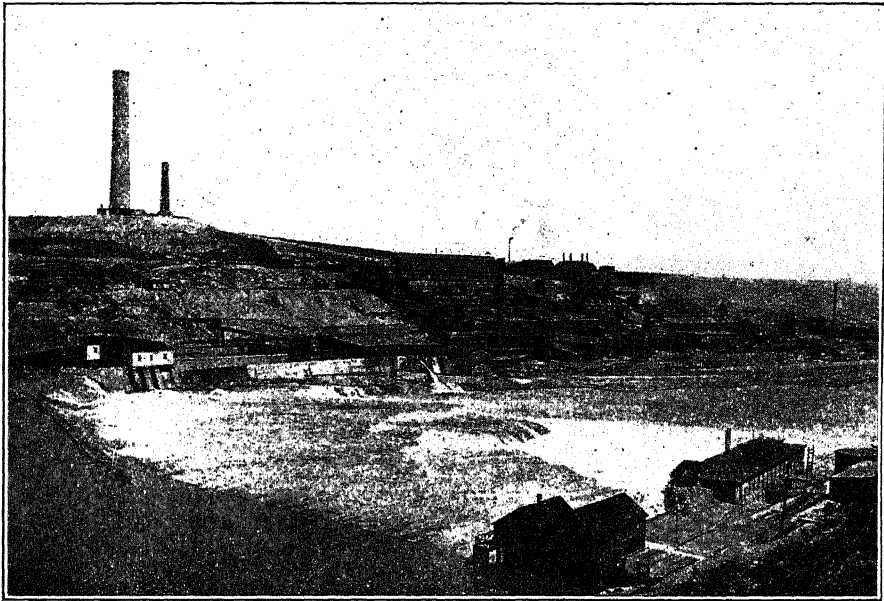
FIG. 6.—BLACK EAGLE DAM AND FALLS.

Black Eagle Plant in foreground. Boston & Montana Smelter across river, and highest stack in the world on top of hill.

These four plants supplying power to the cities of Butte, Helena and Great Falls, and being built at the time when the transmission of power by electricity was in its infancy, formed the nucleus of what has become one of the great power systems of the country.

III. LATER DEVELOPMENTS.

1. Madison River System.

In 1906 and in the succeeding years great strides were made in power development. The Madison river dam was rebuilt and enlarged, and the Madison No. 2 plant installed with a capacity of 9,000 kw. This plant is notable in that it employs a 10-ft. and a 12-ft. wood-stave pipe 7,400 ft. long. These pipes at the time they were built were the largest pipes of this type ever constructed. The Madison No. 1 plant was rebuilt and used as an auxiliary. The voltage of the system was raised to 46,200
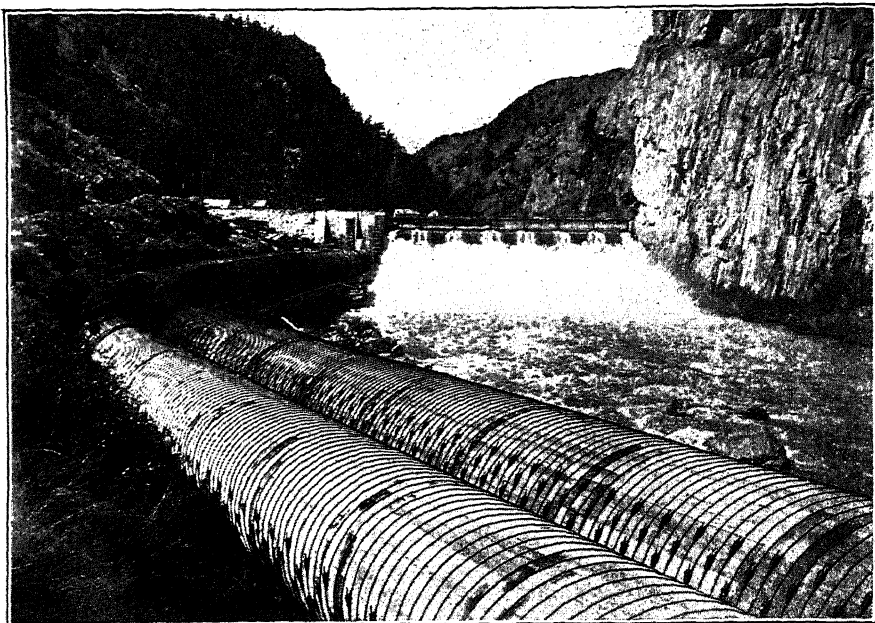
FIG. 7.—MADISON RIVER DAM AND PIPE LINES.
These pipes at the time of their construction were the largest in the country, one being 10 ft. and the other 12 ft. in diameter; length, 7,400 ft.
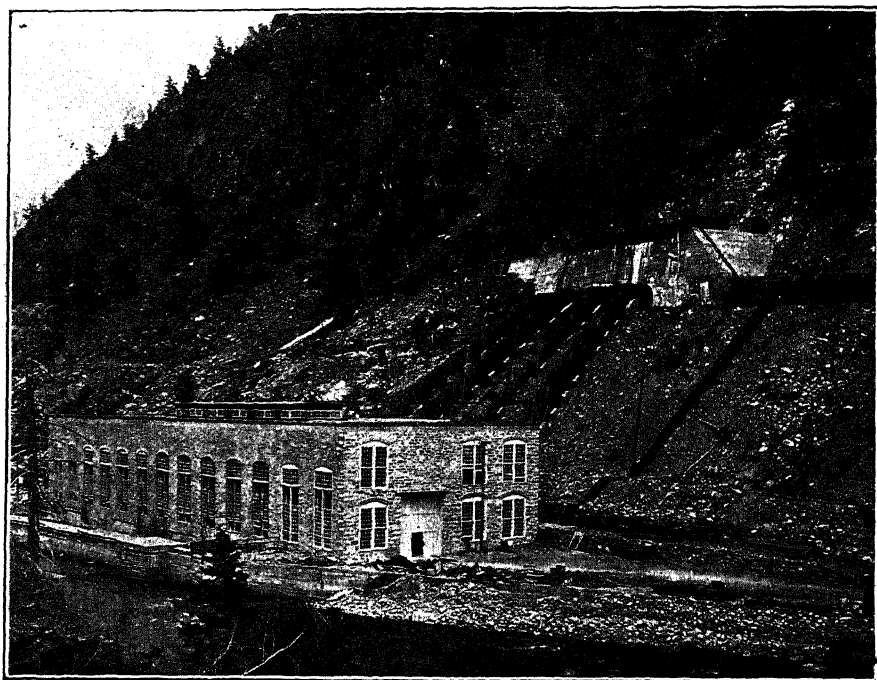
FIG. 8.—MADISON No. 2 PLANT.
9,000 kw. capacity, 110 ft. head.

and additional transmission lines built to Butte, to the Conrey Placer Mining Co. in Alder Gulch, to the Three Forks cement plant and to the towns of Bozeman, Livingston, and later to Billings.

2. Missouri River System.

On the Missouri river below Canyon Ferry was constructed the Hauser Lake plant with a 60-ft. dam and a capacity of 14,000 kw. This plant was connected with the transmission lines to Butte, and the lines were extended to supply power to the Washoe smelter at Anaconda. Numerous branches and extensions were also made between Butte and Helena to supply power to various small mines situated in the intervening district.

3. Great Falls System.

The greatest natural power site in the State, and one of the greatest in the country, is on the Missouri river near the city of Great Falls. In a distance of 8 miles the river falls approximately 400 ft., half of this drop being in abrupt falls, making power development particularly easy and making feasible a total development of about 125,000 kw.

In 1910 the first large development was made at this point. This development was called the Rainbow plant after the falls of the same name. A low diversion dam was built above the falls and the water conducted through two 15 ft. 6 in. steel pipes to a point below the falls where it was utilized at a head of 105 ft. Generating apparatus having a capacity of 21,000 kw. was installed, but upon test it was found that both the water wheels and generators greatly exceeded their original rating, and this plant is now conservatively rated at 25,000 kw.

Power from this plant is transmitted at 6,600 volts to the smelter at Great Falls, and to Butte over duplicate steel-tower transmission lines operating at 100,000 volts. From Butte an additional line is extended to Anaconda so that at present almost the entire capacity of both the Rainbow plant and the Missouri River plants are utilized in the mines of Butte and the smelters at Anaconda and Great Falls. In addition to these main transmission lines additional feeders of smaller capacity have been built from the Rainbow plant to Lewistown, Havre and Cascade.

4. Missoula River Development.

In 1906 and 1907 a timber-crib dam was built on the Missoula river, six miles above Missoula just below the mouth of the Blackfoot river. A 4,000-h. p. installation was made, and power from this development serves the city of Missoula. A high-tension line from this development is also constructed up the Bitter Root valley and serves the cities of Hamilton, Stevensville and Victor.
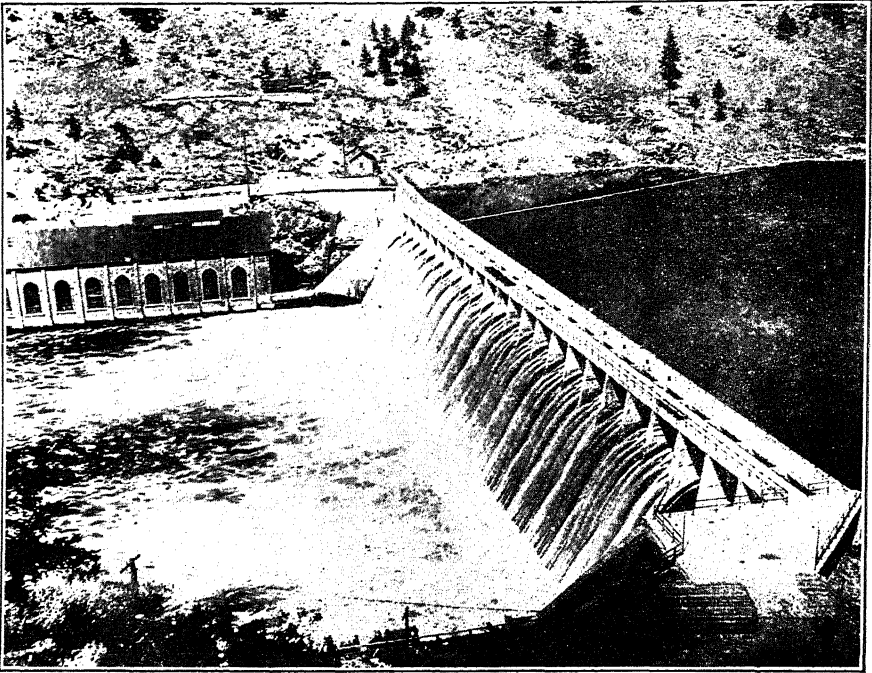
FIG. 9.—HAUSER LAKE PLANT AND DAM.
This is a solid concrete dam 60 ft. high with an adjustable flashboard system for taking care of extreme floods.
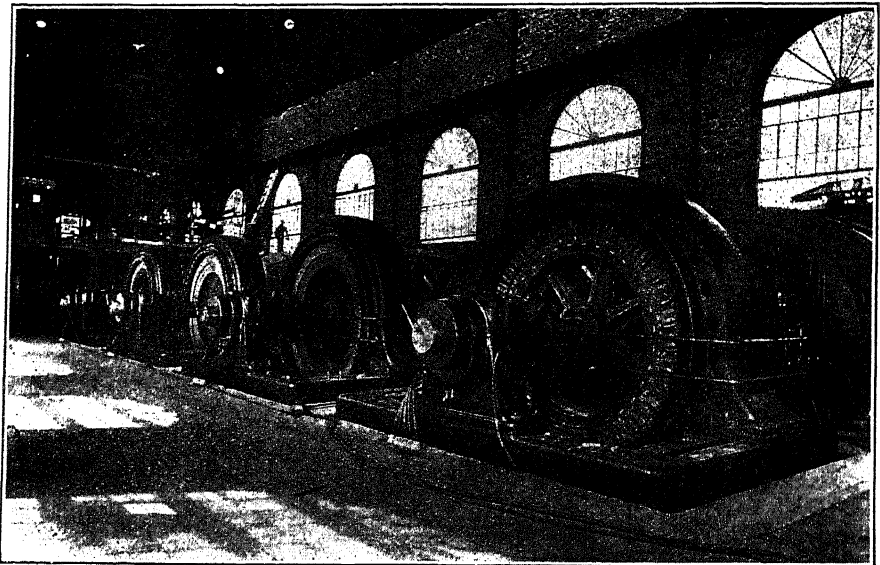
FIG. 10.—GENERATOR ROOM IN HAUSER LAKE PLANT.
Five 2,800-kw. units operating at 65 ft. head.
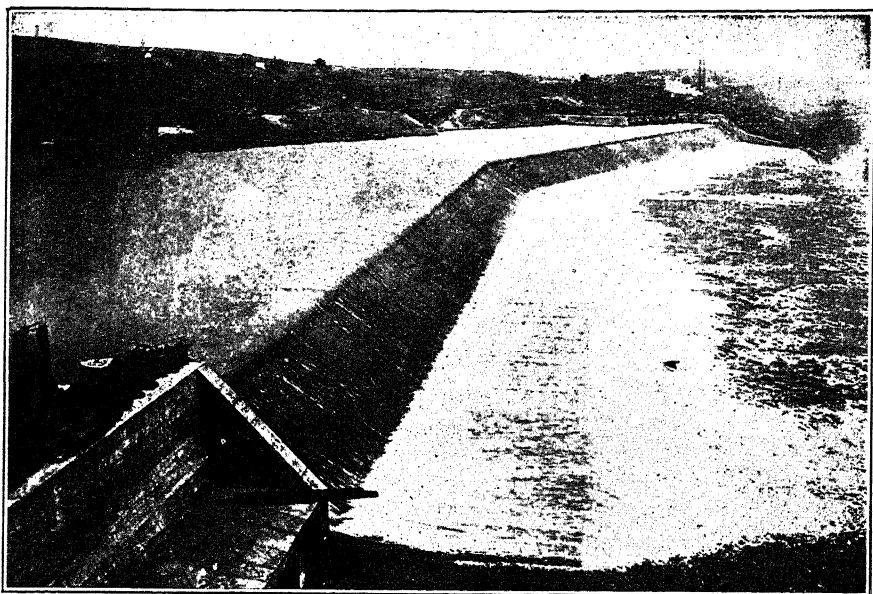
FIG. 11.—RAINBOW DAM SHORTLY AFTER COMPLETION.
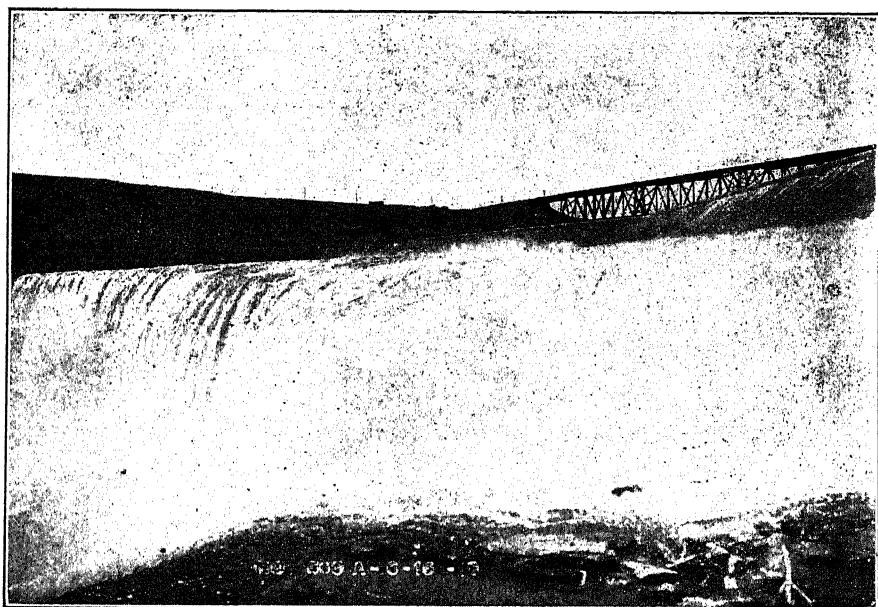
FIG. 12.—RAINBOW FALLS AND DAM DURING HIGH WATER.
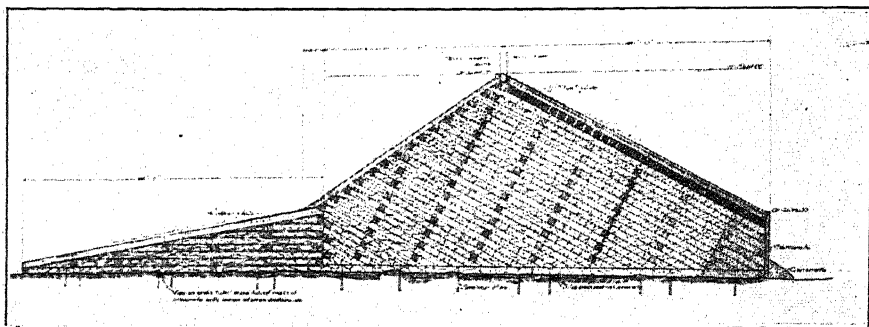
FIG. 13.—CROSS-SECTION OF DAM AT RAINBOW PLANT.
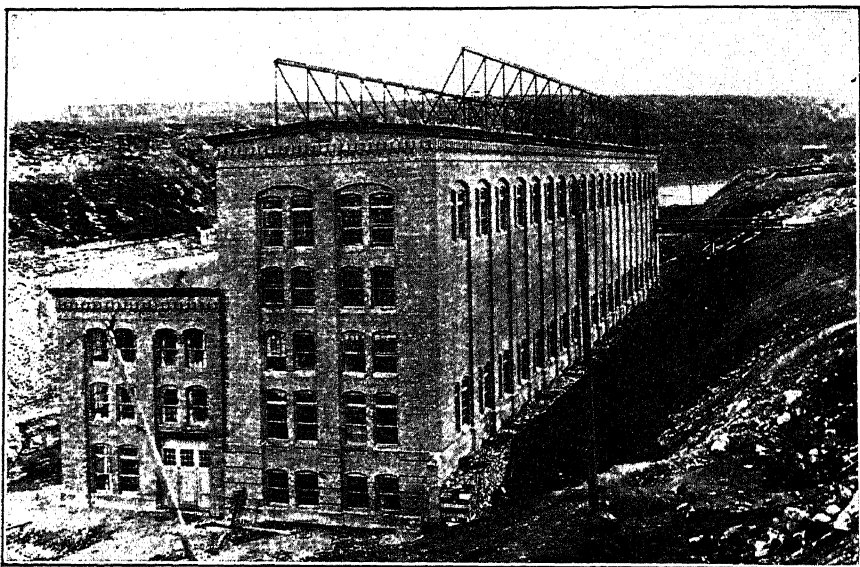
FIG. 14.—RAINBOW PLANT.
21,000 kw. capacity, 105 ft. head, operates at 100,000 volts.
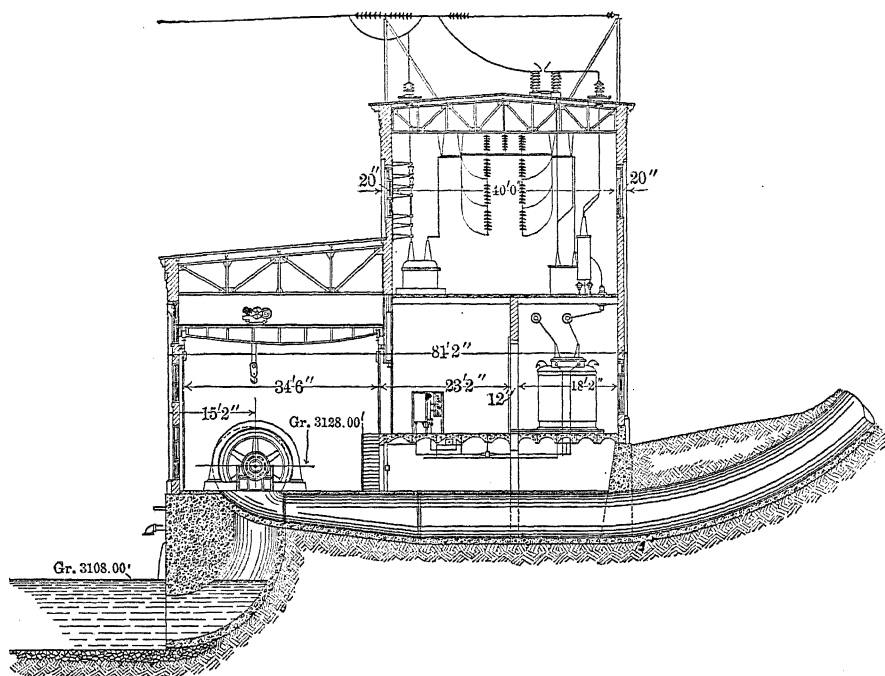
FIG. 15.—CROSS-SECTION OF RAINBOW STATION.
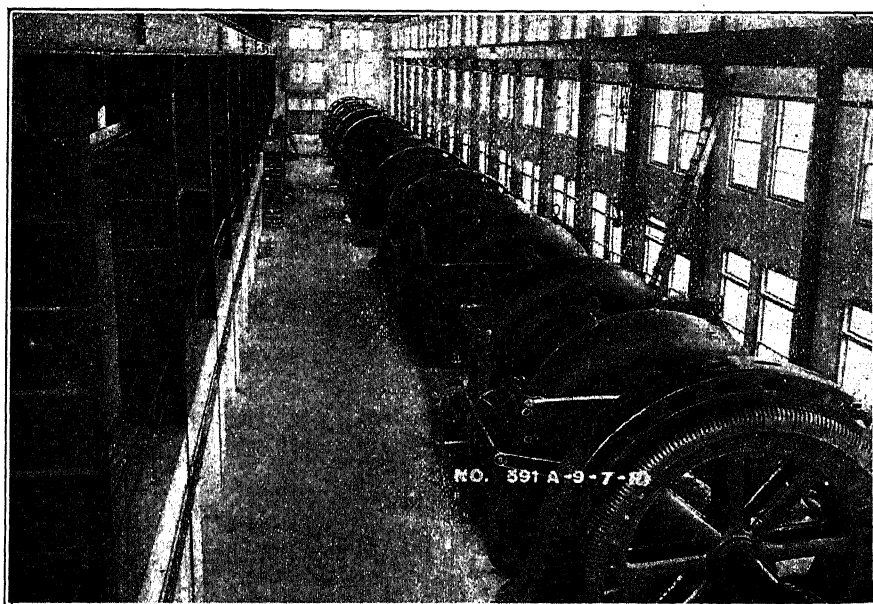
FIG. 16.—GENERATOR ROOM IN RAINBOW PLANT.
Six 4,000-kw. generating units operating at 105 ft. head.

5. *Big Fork Development.*

A 2,500-h. p. development has for some time been in operation on the Big Fork river in the northwestern part of Montana. Power from this development serves the cities of Kalispell, White Fish, Columbia Falls and intermediate points.

In addition to the developments mentioned above there are a number of small powers scattered throughout the State.

IV.   PRESENT CAPACITY OF POWER DEVELOPMENTS.

In addition to the plants previously mentioned there are in operation several smaller plants which are not of sufficient capacity to require
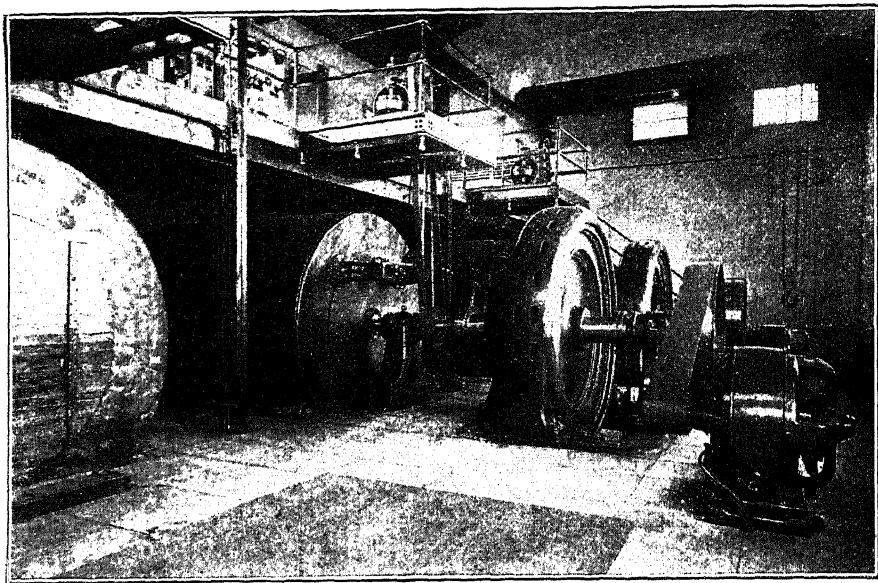
FIG. 17.—POWER PLANT AT BILLINGS.
Three water-wheel driven generators, 360 kw. each.

description. They will, however, be included in the list on the following page, which shows the total normal capacity of all the plants which are now operating on the combined systems of the Montana Power Co. and the Great Falls Power Co.

V.   TRANSMISSION SYSTEM.

The principal transmission lines of the combined system operate at 46,200, 66,000 and 100,000 volts. The various lines are tied together through transformers and operate as one large distributing network into which power is fed from each of the 11 plants and from which power

PLANT	Generating Capacity Kilowatts
Big Hole	3,000
Billings No. 1	1,000
Black Eagle	1,400
Canyon Ferry	7,500
Hauser Lake	14,000
Lewistown No. 1	350
Lewistown No. 2	100
Livingston	1,500
Madison River No. 1	2,000
Madison River No. 2	9,000
Prospect Creek	750
Rainbow	25,000
Missoula River Development	4,000
Big Fork Development	2,500
Total	72,100
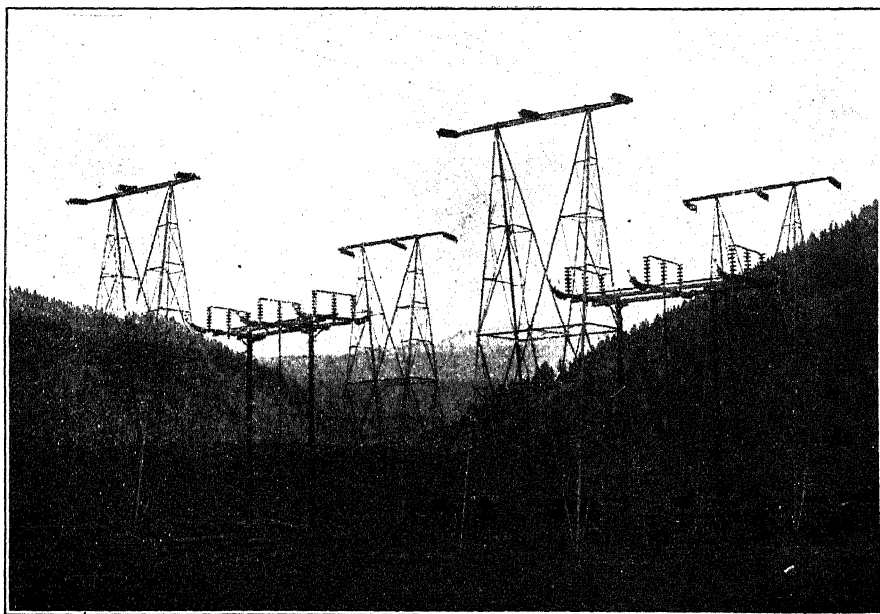
FIG. 18.—SECTIONALIZING SWITCHES ON GREAT FALLS-BUTTE 100,000-VOLT TRANS-
MISSION LINE.
These switches are used to cut the line into short sections to facilitate the location
of trouble.

is taken by about 46 substations, varying in size from the Great Falls Power Co.'s substation at Butte having a capacity of 21,000 kw. down to the lowest standard high-voltage substation of 250 kw.

There is at present in operation a total of 1,300 miles of transmission line of which 340 miles is steel-tower line and the rest is wood-pole line. This transmission system covers a territory approximately 200 miles square.

VI. CHARACTER OF LOAD.

1. Lighting and Small Power Uses.

From this one system, power is supplied for lighting 40 cities and towns in Montana. In addition to lighting, power enters into almost every phase of the industrial life of all these cities and towns. The domestic water supply in all the larger cities of the State is pumped by electric power. Something over 10,000 flat-irons are in use in the homes of the people. Cooking by electricity is becoming popular. Over 500 electric cooking stoves are in daily use. Electrically driven sewing machines and washing machines are in every-day use, and many other devices which tend to lighten the duties of the housewife are operated by electric power.

2. Street Railways.

The street-railway systems of the cities of Butte, Great Falls, Missoula, Helena, Anaconda, Billings and the Gallatin Valley Railway system all receive power from the transmission lines within the State.

3. Mining.

It is admitted that the average cost of steam power for mining purposes in the State of Montana is $125 per horse power per year. In many isolated mining districts this figure rises as high as $150 and even $200 per horse power per year. Electric power is purchased at $50 per horse-power-year in comparatively small quantities, and in large installations prices for power can be obtained as low as $35 per horse-power-year. The advent of electric power in mining districts of Montana has meant that many a property which formerly operated at a loss on a steam basis is now making a profit by the use of electric power.

Although the lighting, street railway and other miscellaneous industries form a very important and necessary. part of the load of the power companies, yet by far the greatest amount of power which is used in Montana is devoted to mining and smelting; in fact, it may be said that

the mining industry and the power industry have developed together, each to a large extent making the other possible.

The extensive use of power for mining makes commercially feasible the development of water power in large amounts and warrants the building of high-capacity, long-distance transmission lines.

This development of power on a large scale is the principal factor in
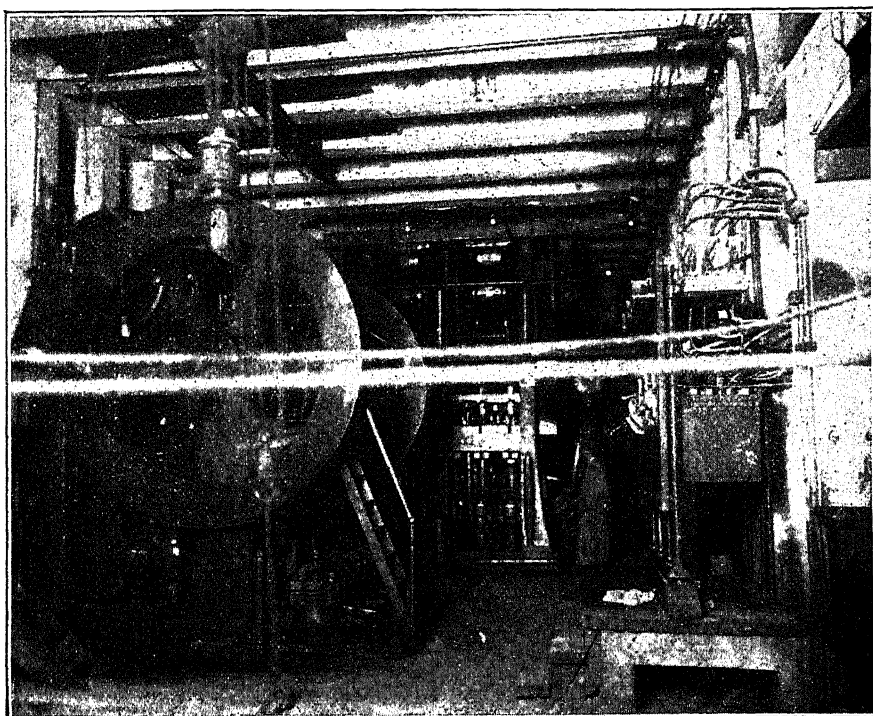
FIG. 19.—1,200-FT. LEVEL UNDERGROUND PUMPING STATION AT LEONARD MINE.

making its cost low. The low cost of power and the low price at which it is sold stimulate its use for all the miscellaneous purposes to which it is adapted. Thus the immense power possibilities of the Missouri river at Great Falls lay dormant until the demand for power in the mines at Butte made it feasible to install a large development and transmission system.

The development having once been established, it proved feasible to continue the extension of lines in all directions from Great Falls, and over these lines power is now supplied for lighting the towns of Havre, Lewistown, Cascade, and a large number of smaller points for operating flour mills, irrigating pumping plants and the coal mines in the Sand Coulee and Stockett district. For this latter purpose alone about 1,200 kw.

are being used; thus it is seen that cheap power in the northern part of the State is made possible by the mining industry in the southern part.

On the other hand the Madison River plant was originally built to supply power in Butte, principally for lighting. The installation of this plant, however, made it possible to deliver power to the Conrey Placer
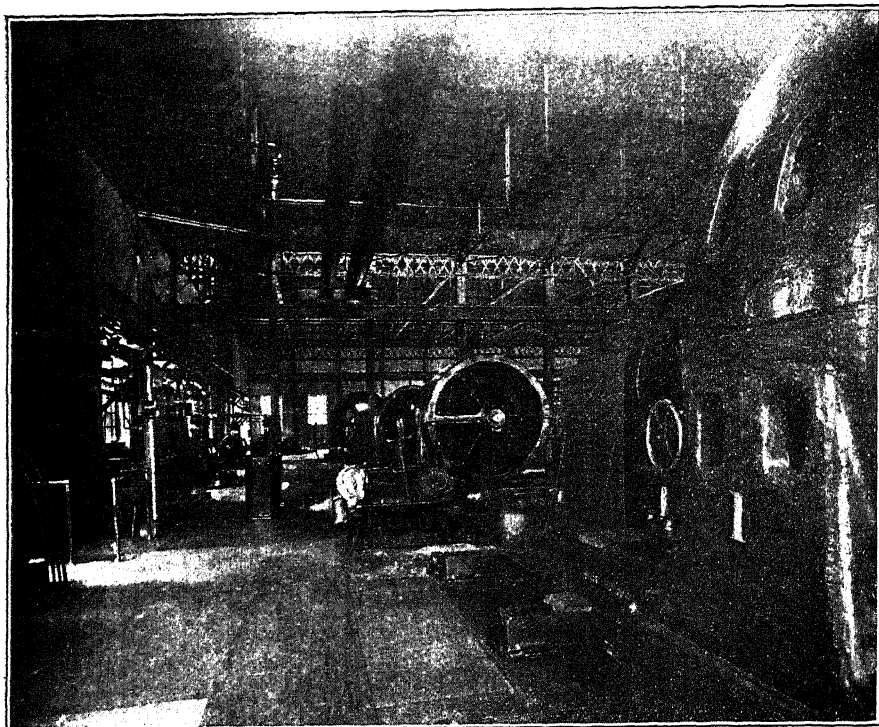
FIG. 20.—COMPRESSOR PLANT AT LEONARD MINE.
Four 600-h.p. motors driving air-compressors.

Mining Co. in Alder Gulch, and made economical the operation of its numerous dredges where the operation by steam was practically prohibitive on account of the high cost of coal and increased amount of labor necessary.

The total amount of power used for mining amounts to 24,000 kw. and for milling and smelting 20,000, a total of 44,000 kw. for the mining industry as a whole. Most of this power is used in Butte, Anaconda and Great Falls.

The mines in Butte use power principally for operating compressors, for hoisting and for pumping. The compressor and pumping load has naturally a high load factor, and therefore forms a desirable load for the power company. The hoisting load has such an extremely low factor

that until very recently it was not considered worth while to operate this load with electric power. To overcome this objection, however, a comprehensive compressed-air system of hoisting has been installed embracing most of the principal mines. Compressed air is supplied to all these mines from a central compressor station which operates in connec-
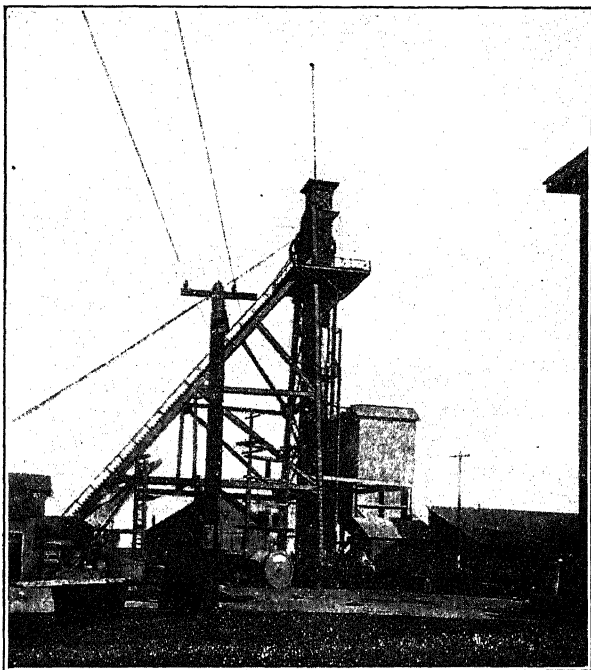
FIG. 21.—HEAD FRAME AT MOUNTAIN VIEW MINE.
Hoisting at this mine is now being done by compressed air from the electrically operated central compressor plant.

tion with a large storage system. The storage system consists of a number of air receivers situated about 200 ft. below the top of the hill upon which stands the compressor plant. At the top of the hill is a water reservoir which is connected to the bottom of the air receivers so that a uniform hydrostatic head equivalent to 90 lb. pressure is maintained on the receivers, and the entire capacity of the receivers can thus be used at full normal pressure, the air being displaced by the water as it is required. When the combined demand of all the hoisting engines exceeds the supply from the compressor plant the excess air is drawn from the storage tanks and water runs down the hill into the tanks to take its place. When the demand is less than the supply from the compressor plant the excess air is forced into the storage tanks, and in turn forces the water back up the hill. In case of a temporary failure of power sufficient storage

is provided to allow the continued operation of the hoists for a sufficient period of time to cover all emergencies.

This compressed-air hoisting system was installed after various electric systems had been considered, and has the following advantages:

First. The total cost of installation was lower than the estimated cost of any other system. This was partly due to the fact that the steam hoists which were already installed were used in connection with the air system, it being necessary to install new cylinders and valve mechanism only.

Second. A larger power-storage capacity is provided than could be economically provided with a straight electric system.

Third. The central compressor plant is operated by synchronous motors, and both the power factor and load factor can be maintained at practically 100 per cent.

Fourth. Excess air from the compressor plant is used to great advantage for supplying the drill system in the mines.

This central compressor plant has a capacity of 7,200 h. p., and it is planned to operate hoisting engines for an aggregate capacity of about 40,000 h. p. That this can be accomplished is due to the fact that each hoisting engine has a load factor of about 5 per cent. and the diversity factor of all the engines taken together is correspondingly low; in other words, when one engine is operating at full load, another may be operating at part load, and several others will be standing idle. To make the combined load as uniform as possible the hoisting schedules are arranged so that ore is hoisted from the different mines at different times of day.

4. Railroads.

The electrification of the Butte, Anaconda & Pacific Ry. is nearing completion. The substation at Anaconda is in operation and locomotives are working successfully at the Anaconda end of the line.

This road has a 26-mile main-line track from Butte to Anaconda and a total of 90.36 miles, the excess being made up in yards and sidings.

For this electrification, 2,400-volt direct current has been adopted, this being the highest direct current voltage thus far used for this class of work. The use of this high-voltage direct current allows advantage to be taken of the light weight and superior operating characteristics of the direct-current motor and at the same time reduces the amount of current to be collected to a point where it can easily be handled by a single pantograph trolley, it also allows placing of substations long distances apart.

This installation will have only two substations, one at Butte and one at Anaconda. Each substation will be equipped with two synchronous-motor generator sets of 1,000-kw. capacity each, giving a total capacity
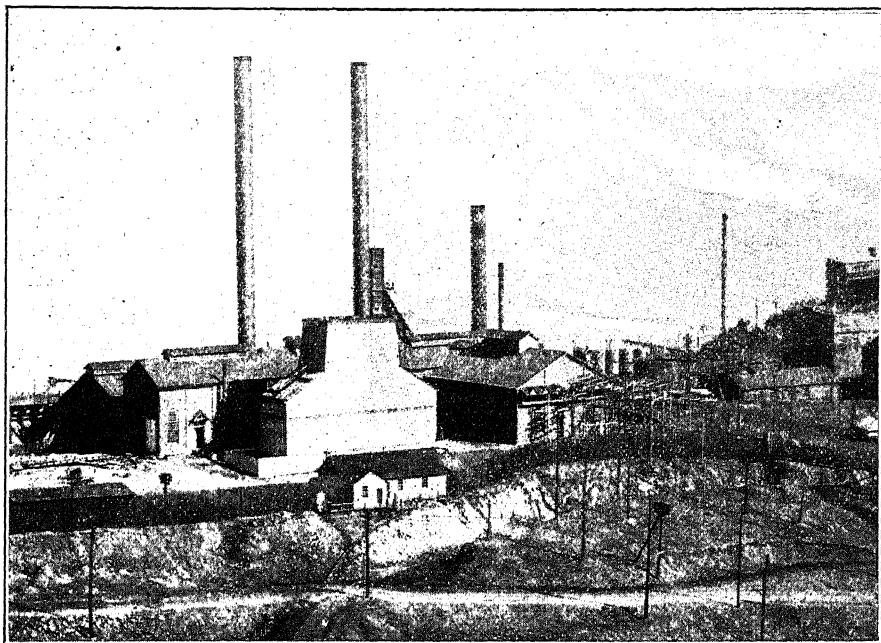
FIG. 22.—60,000-VOLT SUBSTATION AND AUXILIARY STEAM PLANT AT BUTTE.
The substation has a capacity of 10,000 kw. and the steam plant 5,000 kw. Power is
delivered to this station from the Canyon Ferry and Hauser Lake Plants.

of 4,000 kw. to the entire direct-current generating equipment for the
road. The generators have, however, a short time overload capacity
of 200 per cent., so that 12,000 kw. may be supplied as a maximum.

The locomotive equipment will consist of 17 75-ton units. Two of
these units will be operated together for heavy freight service and one
unit alone used for lighter freight and passenger service.

The Chicago, Milwaukee & St. Paul Railway Co. has entered into
contracts with the power companies in Montana to supply power for the
electrification of its line from Harlowtown, Mont., to Avery, Idaho, a
distance of 439 miles of main line with considerable excess in sidings.
Work on this electrification will probably start within the next year or
two, and about 25,000 kw. will probably be required.

It appears to be only a matter of time until all the transcontinental
railways, in their mountain divisions especially, will find it necessary
to turn to electrical operation. This method of operation not only has
the advantage of making large savings in operating expenses on account
of the elimination of coal and the greatly reduced maintenance expense
of electric locomotives as compared with steam locomotives, but also on
account of the fact that with the increase in traffic the capacity of the
whole line is limited by the tonnage which can be hauled over the steep

mountain grades.   Inasmuch as electric locomotives can be made with a much greater continuous power capacity than any steam locomotives yet built, it follows that trains can be hauled over the heavy grades at greatly increased speeds and the tonnage hauled over the division increased
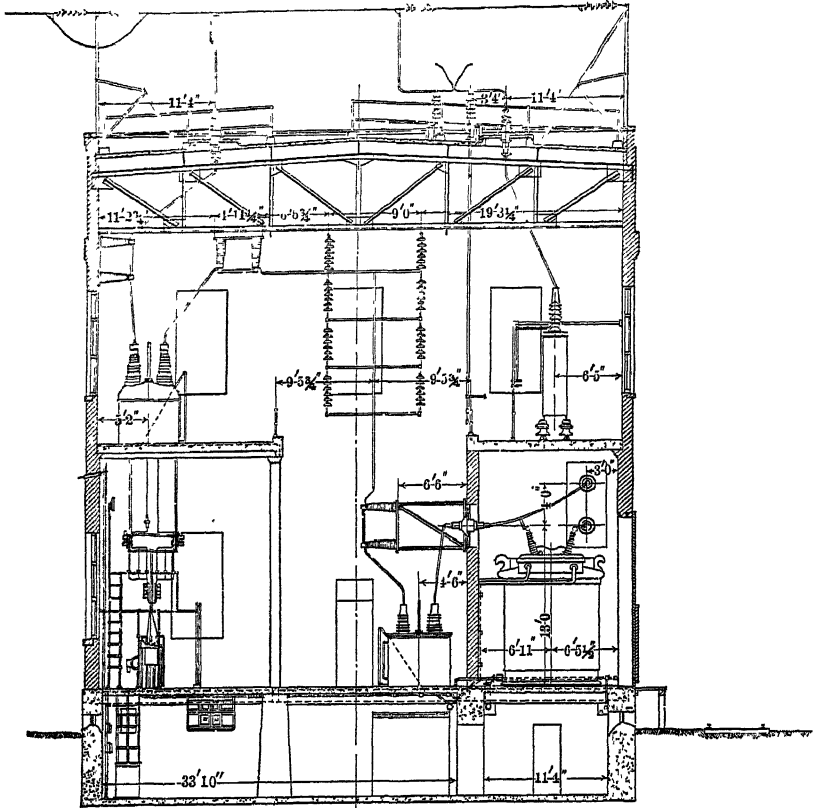
FIG. 23.—CROSS-SECTION OF 100,000-VOLT SUBSTATION AT BUTTE.
Present capacity 21,000 kw.

accordingly, while to get a corresponding increase with steam operation would require double tracking the road.

5.   Irrigation.

The average value of the agricultural land in Montana is increased tenfold by irrigation.   Up to the present time irrigation has been practiced for the most part only where the natural conditions were such that it was easy to get water on the land by gravity ditches.   Some of the best land in the State, however, is so situated that it is impossible to supply it with water by gravity and pumping must be·resorted to, in fact in a great many instances it is cheaper to pump the water than to pay fixed charges

on the necessary canals, ditches and pipe lines required to supply the water by gravity.

During the past year the first large pumping plant for irrigation was installed in the Prickly Pear valley near Helena by the Montana Reser-
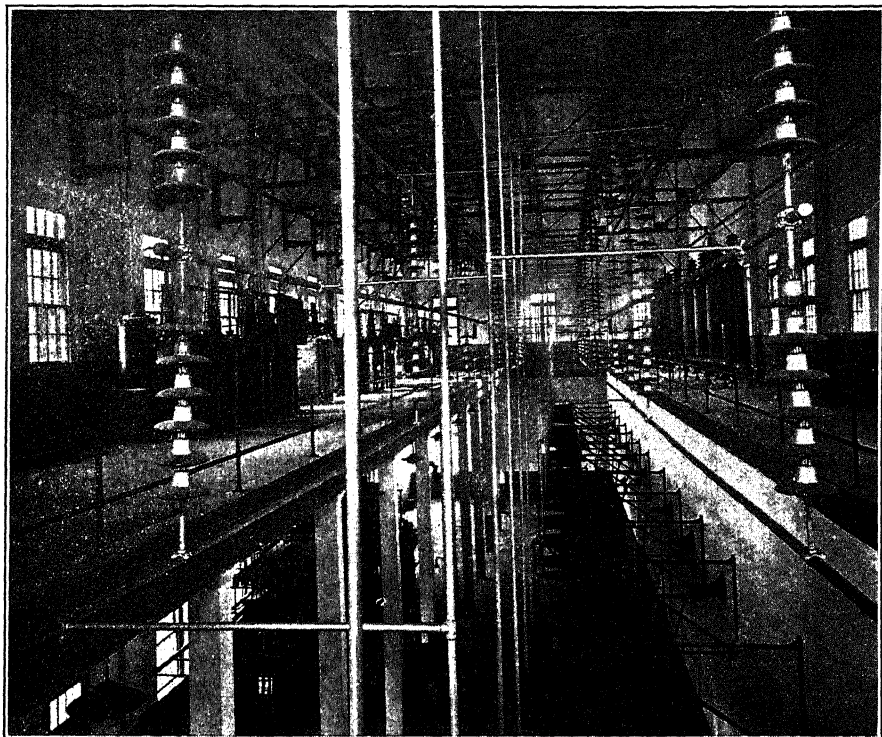
FIG. 24.—BUSBARS AND SWITCHES IN 100,000-VOLT SUBSTATION AT BUTTE.
The high tension connections in this station are made of ordinary iron pipe.

voir & Irrigation Co. This plant during the present season will supply water to 5,000 acres, and will require 1,800 h. p. for its operation. This plant forms the first unit of the Prickly Pear irrigation project, and as fast as the land becomes settled two additional units of about the same size as the first will be installed, making a total of 5,400 h. p. for this one project.

Near the city of Billings the Lockwood irrigation project is being established with a pumping plant of 500 h. p.

In the Sun River valley, west of Great Falls, the government has started work on what will be the largest irrigation project in the State. A gravity system for supplying water will be used, but for building the dams and canals the United States has contracted with the Great Falls Power Co. for a maximum of 3,484 h.p. for five years.

To supply water during the irrigation season the Montana Reservoir & Irrigation Co. is building a dam to form a storage reservoir near the head waters of the Madison River close to Yellowstone Park. This reservoir will have a capacity of 15,000,000,000 cu. ft., which will be sufficient to store the entire spring and summer flood of the Madison river. This water will be available for use throughout the entire length of the Madison river and in the Missouri river down to Great Falls, or even farther. The stored water will be used in two ways: First, for pumping on to land for irrigation, and second, for the generation of additional power in the present power plants, this power to be used for operating pumps. In this way the water is made to pump itself into the land.

VII. Operation of System.

The operation of the power system composed of many plants operating in parallel is somewhat more complicated than the operation of a system having only one generating system. There are also many advantages in the operation of such a combined system which cannot be realized in the operation of a single plant. With a single plant the maximum amount of power which can be sold is ordinarily limited by the amount which can be generated at the time of minimum flow of the river upon which the plant is situated. This minimum flow can be increased by the building of storage reservoirs, but storage reservoirs are expensive and in many instances would not pay where the water is to be used by a single plant.

In the case of the power system in Montana the Hebgen reservoir of the Montana Reservoir & Irrigation Co., previously referred to, is so situated that its stored water will be utilized in turn by the two Madison River plants, by the Canyon Ferry and Hauser Lake plants and by the three Great Falls plants, being a total of seven plants, each of which will derive full benefit from the storage.

It also occurs in the development of water power that some plants are well adapted to operate on a variable load while others can only operate to advantage on a uniform load. Thus a development in which the head is produced by means of a dam ordinarily has a large pond from which large quantities of excess water can be drawn for short periods of time to supply a temporary demand for power much in excess of the average plant output. This type of development ordinarily employs very short pipe lines so that the cost of water conduits for this excess use of water is not great. This type of development when connected to a large system is called a peak-load plant.

Another type of development consists of a low diversion dam and a long flume or pipe line which conducts the water to a point further down

the river where the plant is situated. With this type of development the largest item of cost is the pipe line, and inasmuch as the low diversion dam creates very little storage, and as the cost of development would be greatly increased if the pipe lines were made large enough to carry more than the normal flow of the river, this type of development is best adapted to operate at a uniform and steady load. A development of this kind is called a base-load plant.

A base-load plant operating alone to supply power for average requirements must necessarily waste a large quantity of water on account of the fact that due to the variable demand for power it cannot operate at full load continuously, and during the periods when the demand for power is light, water is wasted over the dam. On the other hand, the average peak-load plant will have a greater storage capacity than can be well utilized by the usual variations in load, which means that a certain development if it were to operate alone could be made for, say, 10,000 kw., while it might, if operated in connection with a base-load plant, be made for a development of 15,000 kw. merely by the addition of extra generators and water wheel. In this type of development, where the dam is usually the greatest item of cost, the additional power obtained only by the addition of generating units is obtained at a very low cost for the additional horse power.

Where one plant is the sole source of supply for a system every precaution must be taken to maintain continuity of service. A large number of generating units must be installed so that if one unit is out of commission for repairs the station will be reduced only a small percentage. Where the system is supplied by many plants the shutting down of a unit in any one plant is of less consequence as it reduces the capacity of the system in a percentage corresponding to the ratio between the capacity of one unit and the capacity of the entire system. For this reason the generating units in such a plant can be made larger, which will result both in a lower first cost and a higher efficiency.

When plants are situated on different rivers, which take their water from different water sheds, the minimum flow on one river will not correspond in time with that on another. For this reason when one plant has its output reduced by extremely low water another plant will have plenty of water and will be able to carry a part of the first plant's load. Also, even with plants at different places on the same river, one plant may have its output temporarily reduced by ice troubles while another plant farther down the river will be free from such trouble, and can carry an extra share of load to make up for the deficiency at the first plant.

The balancing of the variations in river flow among different plants will be carried on with great advantage between the plants on the Missouri river and on the Clarks fork. These rivers flow on opposite sides of the

Continental Divide, in parts of the State, which, although only 200 miles apart, have very different climates. The variations in the flow of the two rivers are, therefore, such that with the two developments tied together by long-distance lines power can be continually shifted back and forth from one system to the other, allowing each development to utilize the total flow of its river to the best advantage.

It is estimated that the advantage to be derived from the combined operation of many plants due to the causes above enumerated amounts to at least 50 per cent. in the economical utilization of the total available power.

To give an idea of the value of the power which is being generated by the plants of the system under consideration, which are now producing 49,500 kw., it may be stated that to produce this amount of power from coal would require the yearly consumption of 750,000 tons; at the average price of $4, which would apply at the points where the power is used, this would amount to $3,000,000 per year.

When it is remembered that this value in coal is, by means of water power, being saved for future generations, and that during the next 10 years this saving will in all probability be increased about six or seven times, it is not difficult to appreciate that true conservation consists in encouraging rather than restricting the free development of water power.

VIII. Increase in Use of Power.

The use of electric power is increasing by leaps and bounds. The power companies must maintain a sufficient reserve capacity in power plants to accommodate increases in the demand for power as they occur. It becomes a difficult matter to predict far enough in advance the increase in load which will have to be taken care of. Such a prediction can be based on little else than past experience.

IX. New Power Developments.

To take care of the general increase in load, especially the Chicago, Milwaukee & St. Paul railway electrification, work has been commenced on two new, large developments, one at the Great falls of the Missouri and one at Thompson falls on Clarks fork. The Great Falls development will have a capacity of 60,000 kw. and the Thompson Falls development a capacity of 30,000 kw.

The Great Falls development is situated about 8 miles down the river from the city of Great Falls, and utilizes a natural fall in the river of about 85 ft., which fall will be increased to a total of 150 ft., by the erection of a solid masonry dam immediately above the falls. The dam will have a crest line 1,000 ft. long and will have an overflow section designed

co take care of the maximum flood of the river, estimated at 100,000 sec.-ft. Generating equipment will be installed to utilize the normal flow of 6,000 sec.-ft. in six vertical single-runner water-wheel units of 10,000-kw. capacity each. To illustrate the high efficiency which can be obtained in a modern water wheel of this type, it may be stated that manufacturers have promised an efficiency of 90 per cent. for these wheels, and this efficiency has even been exceeded by similar wheels which have already been built. The power plant will be built on a rock ledge just below the falls and will take its water through six short steel pipes leading directly to the dam.

The Great Falls development will supply power to the territory covered by the present system and to the Chicago, Milwaukee & St. Paul railway from Harlowtown to Deer Lodge.

The Thompson Falls development will be similar to the Great Falls development in that approximately one-half the head exists in the form of a natural fall and the other half will be developed by a dam immediately above this fall. In the case of Thompson falls, however, the plant will be situated some little distance below the falls and the water conducted thereto through an old dry channel which will be flooded when the dam is built. The design of generating units and the general plan of the power house will be similar to that at Great Falls. A total head of about 50 ft. will be utilized.

The Thompson Falls development will supply power to the territory west of that which is covered at present and to the Chicago, Milwaukee & St. Paul railway from Deer Lodge to Avery, Idaho.

The addition of these two plants will more than double the present capacity of the system, but it is estimated that this increase will be absorbed within a very short time after the completion of the plants, which will require two or three years.

X. UNDEVELOPED POWERS.

In addition to the plants already built and under construction there are a large number of power sites standing ready to be developed as demands for power continue to increase. Although these developments are situated in all parts of the State, the larger possibilities exist in the northwestern part of the State where developments like the Flathead lake, Kootenai falls and other large streams present possibilities for the development of at least 250,000 h. p.

DISCUSSION.

A MEMBER:—Have there been any definite studies made in the hydrology of this system? That is to say, what is the total extent and area of the drainage or catchment basin above these plants? What is the total annual precipitation, and what is the percentage of run off? That is a very interesting question to engineers and a question engaging a great many, because, unfortunately, there is not the amount of information on that subject which there ought to be.

FRANK SCOTTEN, Great Falls, Mont.:—There are a great many figures on the subject, but I am not familiar enough with them to quote from memory. The entire drainage area has been figured in unit feet, and the precipitation is well known, but I am not familiar enough with it to quote.

PROF. JOSEPH W. RICHARDS, South Bethlehem, Pa.:—I think this paper is a very fundamental one, because it goes right to the root of the future development of Montana as a State, and of its industries. It is exactly in line with what is being done in other parts of the world where coal is scarce and where water powers are plentiful, and I see in it the promise and the prospect of great development which can put Montana ahead of perhaps many other States which have good supplies of coal. I would like to call further attention to one detail of the use of this power, which is, that when a large amount of it is in use for traction and lighting, there are necessarily peaks (speaking in the electrical sense) where the power is used heavily, and deep valleys where the power is used only lightly. You should therefore get from the electric power companies power at certain parts of the day and in certain parts of the year very much cheaper than if you use it continuously. In the East we are finding considerable uses for cheap power used in this way during limited periods.

I take it for granted that in these mining States you have a large use, for instance, for steel castings. I am told that you make very few in Montana. In the East we have electric furnace plants which are using power in the "valleys" of the power supply companies and are making steel castings at a considerable profit, although they are paying as much as $60 a horse power-year for their power. This is just one example of how you can build up your own industries, help yourselves by having your own supply of necessary materials for the mining industry, and save money from going out of the State. This is only one of a dozen or more electro-metallurgical or electro-chemical industries which might be of direct use to your people and which certainly could be established here within a short time.

Use of Electricity in Mining in the Butte District.

BY JOHN GILLIE, BUTTE, MONT.

(Butte Meeting, August, 1913.)

PRIOR to the year 1902 electricity was used in the Butte district only for lighting, for the tramming of ores on the surface, and for the electrolytic refining of copper. In that year the Canyon Ferry hydro-electric plant on the Missouri river, with a transmission line to Butte and Anaconda, was completed, and enabled the mining operators to purchase electric current in large quantities at reasonable rates, and its application and use in the various mining operations and reduction works proceeded rapidly.

The principal applications of electric current in metal mining may be listed as follows:

Compressing air for hoisting.
Compressing air for rock drills and pneumatic tools.
Pumping.
Ventilating.
Lighting.
Tramming.
Compressing air for blast furnaces and converters
Running shop and miscellaneous motors.

Compressed Air for Hoisting.

The Anaconda Copper Mining Co. operates 22 mine shafts and plants at Butte. Of these, 15 are large plants and hoists, through which most of its ores are hoisted.

In 1912 this company commenced using compressed air in place of steam, in operating a number of its large hoists, and at this date has 10 main hoisting engines and 10 auxiliary hoists driven by compressed air.

The compressed air is furnished from a central plant containing six compressors, each having a capacity of 7,500 cu. ft. of free air per minute compressed to 90 lb. pressure in large receivers, and conducted by suitable pipe lines to the various hoists, the air being reheated before passing to the engines. Each of these compressors is driven by a 1,200-h.p. synchronous motor, directly attached to the drive shaft, and takes electric current at 2,200 volts. Two additional compressors are being installed,

making eight in all, which will complete the compressing plant. Seven of these machines will run continuously, and they will use 8,000 e.h.p. The excess air, or blow-off, goes into the rock-drill air system.

The hoisting engines, as formerly used with steam, had cylinders 30 or 32 in. in diameter and 72 in. stroke. To adapt them to the use of compressed air, the old cylinders were replaced with new ones, 34 in. in diameter, having valve mechanism of the most approved type.

Compressed Air for Rock Drills.

The air in rock-drilling operations is used at 85 lb. pressure, and for the large operations is usually furnished from three or four central plants, the smaller mines using individual machines. A total of 13,455 e.h.p. is used for this purpose, and the voltage used is from 440 to 2,200.

Pumping.

In draining the mines electrically driven pumps are very generally used, and a total of 4,780 e.h.p. is necessary. The large pumps are quintuplex vertical plunger type, with a capacity of 600 gal. against a head of 1,200 ft. The smaller units are of various types. Centrifugal pumps are not used on account of the acid nature of the mine waters.

Ventilating.

The mines of the Butte district are warm, due to the oxidizing of the sulphide minerals, and forced ventilation is largely resorted to. Twenty-five hundred electric horse power is used for driving ventilating fans. In addition to fans and blowers, compressed air is used after blasting to clear the mines of powder gases and foul air.

Tramming.

Electric locomotives are being introduced in all tramming, both underground and surface, and this work requires 1,860 e.h.p. Direct current 500 volts is used for surface tramming; D. C. 250 volts is used for all underground tramming. There are a few storage-battery locomotives in use at Butte, but the general system is trolley, using D. C. 250 volts. The mine locomotives are 18-in. gauge, weight 3 tons, have two 10-h.p. motors, and a draw-bar pull rated at 1,200 lb. The average load hauled per trip is 12 tons, including weight of cars and ore, on the usual mine track with grade of 0.5 per cent.

Lighting.

Underground, the stations, main cross-cuts and drifts are electrically lighted, as well as the surface plants and property. A total of about 1,300 h.p. is used for this purpose.

Ore-Reducing Works.

At Butte, Anaconda, and Great Falls, all the Butte ores are treated. In the case of zinc ores, concentrates containing about 50 per cent. of zinc are produced, and these are shipped to the natural-gas districts for smelting and recovery of the metals.

For copper ores about 80 per cent. is reduced to the extent of making converter or blister copper, containing the gold and silver, which for further refining and separation is shipped to the large electrolytic refineries on the Atlantic coast. The remaining 20 per cent. is fully treated or reduced, and merchantable wire bars and other forms of finished copper are produced, and the gold and silver is also separately obtained. This is done at an electrolytic refinery at Great Falls. In these ore-reducing plants 24,177 h.p. are used, in driving concentrators and blowing engines, and in operating cranes, shops, and the different power machinery in connection therewith.

Other Applications of Electricity.

In connection with the use of electricity in mining, it might not be out of place to briefly mention the electrification of the Butte, Anaconda & Pacific railway, which is now nearly completed. This railroad connects the mines at Butte with the reduction works at Anaconda, and handles about 5,000,000 tons of freight yearly, consisting of ores from the mines and supplies of all kinds to both mines and smelters. In addition to the freight hauled, the passenger business between the two cities is taken care of by this road. The conditions, including grades, curvature and climate, are sufficiently severe to thoroughly prove this method of railway transportation.

The constructing engineers have adopted for this work a 2,400-volt D. C. system, with overhead trolley. Freight locomotives will consist of two units, each weighing 80 tons. Passenger locomotive will be one 80-ton unit geared higher to permit of a speed of 45 miles per hour. A part of the road has been electrically operated for several weeks, and is working very satisfactorily.

Near the famous old placer camp of Virginia City, Mont., about 90 miles southeasterly from Butte, four large placer dredges are operated by electrically driven motors, and are proving an entire success.

Not all of the applications of electric energy to mining have been given here, but enough to show that where electricity can be procured at proper prices and conditions it is a most valuable aid to both miner and metallurgist in effecting economies in their business.

The Electrification of the Butte, Anaconda & Pacific Railway.

BY R. E. WADE,* ANACONDA, MONT.

THE Butte, Anaconda & Pacific electrification is of peculiar interest, in an incidental way, to the entire mining fraternity, and especially the engineering branch, not only in this great Northwest country but throughout the entire Western mining country, which is rich in minerals and water power. It is probably fair to say that the prime object of mining and smelting engineering is to reduce the cost of production, and in the case in hand the relative location of mines and smelter brings the cost of transportation of ore to the front. While some might assign this item to the railway people, it is inseparably a part and parcel of the general scheme of the production of metal from the ore. In considering the electrification of the B., A. & P. railway the mining and smelting engineer, above all, wants to know what saving will be effected by the change in motive power. Unfortunately the new power has been in use during a very limited period and on the Smelter Hill section of the road only, and no reliable data, from actual service conditions, can be obtained until the entire electrified section has been in service for a reasonable length of time. Any remarks on this subject at the present time, therefore, must of necessity be general in nature and more or less descriptive.

The railway, electrical and mining worlds and the country in general are indebted to the B., A. & P. railway for the pioneer work it is now doing, as it undoubtedly marks the beginning of the electrification, within the comparatively near future, of all transcontinental roads between the Missouri river and the Pacific coast. Together with this goes the development of an untold wealth in water power and all natural resources within the same territory.

For the first time in the history of electric railway work, in this country, direct current at a potential of 2,400 volts is being used. With this potential there is a great saving in transmission of energy to the locomotives and there are only two substations, one at either end and 26 miles apart, for supplying the power for the movement of trains of heavy tonnage over the main line and for switching movements at terminal and intermediate yards, and comparatively little feeder copper is required.

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* Non-member.

The section of line that is being electrified lies between the receiving and distributing yard on Butte Hill and the concentrator bins at the Washoe smelter, Anaconda, together with the yards and tracks in the vicinity of the Butte passenger station, the Anaconda station and shops, the various departments of the smelter and the intermediate yards and sidings. It comprises about 30 miles of main-line single track, which, together with the various yards and sidings, makes up a total of about 90 miles on a single-track basis. The haulage of copper ore from the Butte mines to the smelter at Anaconda, which is the principal traffic, together with mine supplies, lumber, etc., moving in both directions, amounts to practically 5,000,000 tons of freight per year, or an average of 13,700 tons daily. The freight trains on the main line, weighing 3,400 tons and made up of 50 loaded ore cars, will be handled against a ruling grade of 0.3 per cent. Those on the Smelter Hill line, weighing from 1,350 to 1,400 tons and made up of 20 or more loaded ore cars, against a grade of 1.1 per cent., and those on the Butte Hill line, weighing from 750 to 800 tons and made up of empty ore cars and loaded supply cars, against a grade of 2.5 per cent., will be handled by locomotives consisting of two units. Single units will be used for making up trains in the yards and for spotting cars. Two of the units coupled to a loaded ore train are shown in Fig. 1, which also shows the overhead construction.

It may be interesting to note, as a matter of comparison between steam and electric motive power, that whereas in transporting ore between the East Anaconda yard and the concentrator bins, a distance of 7.25 miles against a grade of 1.1 per cent., the heavy steam locomotives require from 50 to 55 min. to make the trip with 16 cars of ore, a pair of electric locomotive units is making this same trip in 24 min. with 20 car loads of ore. In other words, the electric locomotive handles 25 per cent. more tonnage and consumes about one-half the time required by the steam locomotive.

The initial equipment consists of 17 locomotive units, 15 for freight and two for passenger service. Each weighs approximately 80 tons. When two units are coupled together they are operated in multiple from one master controller and they will haul the 3,400-ton train at a maximum speed of 15 miles per hour against the 0.3 per cent. grade and at 21 miles per hour on tangent level track. The same two units will haul the 1,400-ton train against the 1.1 per cent. grade at 17.5 miles per hour. The passenger locomotives are the same design as the freight locomotives except that they are geared for a maximum speed of 45 miles per hour on tangent level track. The

average passenger train is composed of one locomotive unit and three standard passenger coaches.   The general design of the locomotives is of the articulated double-truck type, all weight being on the drivers. The cab is of the box type, extending the entire length of the loco-
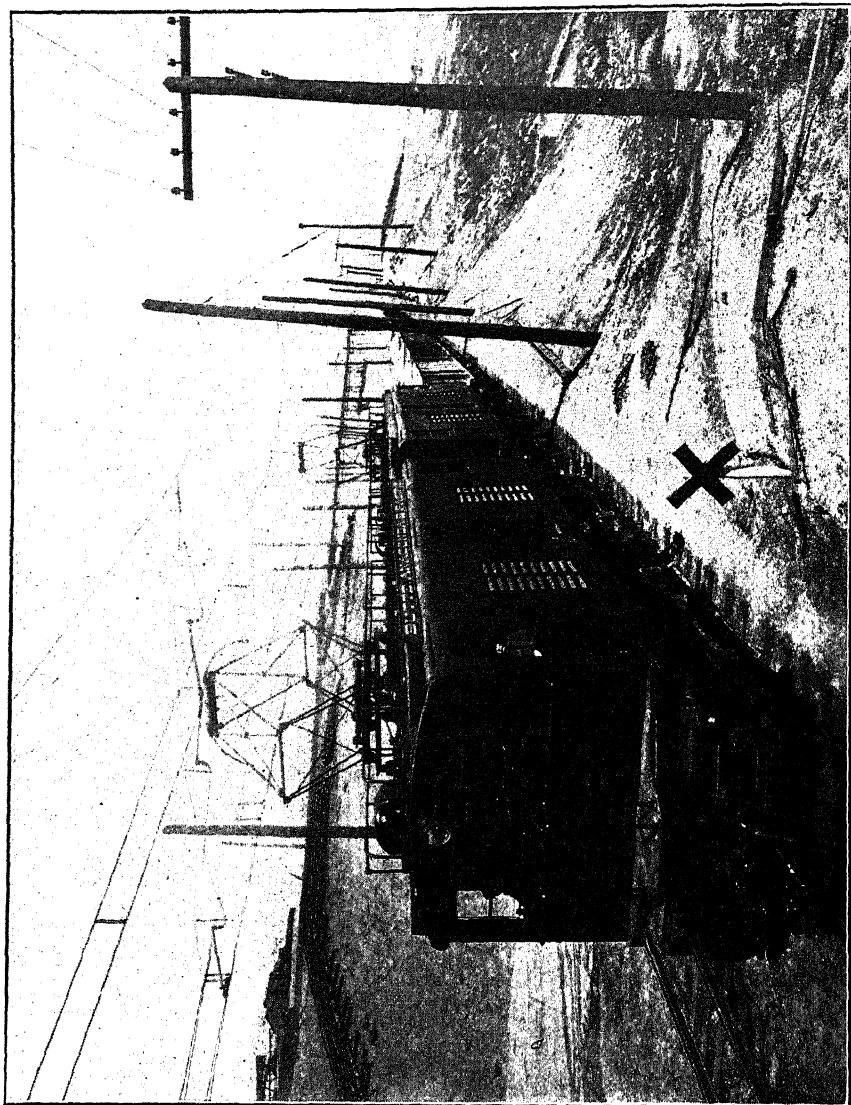
FIG. 1.—ELECTRIC LOCOMOTIVE IN USE ON THE BUTTE, ANACONDA & PACIFIC RAILWAY.

motive, and is of steel construction throughout.   It is divided into three compartments, the center compartment containing the control apparatus, air compressor, and dynamotor, the latter for furnishing 600 volts for lights, air compressor, and operation of control circuit.

The two end compartments are for the accommodation of the engineer and contain the usual master controller, brake valves, and sand boxes. In addition, there is apparatus for ringing the bell and sanding by compressed air, and raising and lowering the pantograph trolley.

The principal data and dimensions applying to the locomotives are as follows:

Length inside of knuckles	37 ft. 4 in.
Height with trolley down	15 ft. 6 in.
Width over all	10 ft. 0 in.
Total wheel base	26 ft. 0 in.
Rigid wheel base	8 ft. 8 in.
Total weight	160,000 lb.
Wheels, diameter	46 in.
Tractive effort at 30 per cent. co-efficient	48,000 lb.
Tractive effort at one-hour rating	30,000 lb.
Tractive effort at continuous rating	25,000 lb.

The motors are of the GE-229-A commutating-pole type, wound for 1,200 volts and insulated for 2,400 volts. A forged pinion is mounted on each end of the armature shaft and meshes into a corresponding gear mounted on the wheel hub, providing a gear reduction of 4.84 on the freight locomotives and 3.2 on the passenger locomotives. The motor is designed especially for locomotive service, is inclosed, and provided with forced ventilation. Air is circulated over the armature and field coils and over and through the commutator, through longitudinal holes in the armature core, and thence exhausted through openings in the bearing head. The continuous capacity of each motor is 190 amperes on 1,200 volts under forced ventilation, and the input is 225 amperes on 1,200 volts for the one-hour rating. For the double unit the continuous rating is equivalent to an output of 2,100 h.p.

Current is collected by overhead trolleys of the pantograph type. They are pneumatically operated, and can be put into service from either engineer's compartment by hand-operated valve. The current collector proper consists of a roller made of steel tubing 5 in. outside diameter by 24 in. long, instead of the fixed or sliding pan usually employed for this purpose with the pantograph trolley. When two units are coupled together the trolleys are connected by couplers between the two units so that current may be obtained from both trolleys or from a single trolley, as may be desired.

The passenger coaches are to be heated by hot air, the air being forced through electrically heated coils by a motor-driven blower, 2,400-volt current being used for both heating coils and blower motor. The current for lighting passenger coaches will be taken

from the dynamotor in the passenger locomotive at a pressure of 600 volts.

The apparatus in each of the two substations consists of two motor-generator sets, each consisting of a synchronous motor directly connected to two 1,200-volt D. C. generators, the latter being connected in series to give the 2,400-volt line potential, together with two motor-driven exciter sets and the necessary switch board panels and control apparatus, most of which has been specially designed for this work.

Within the last few years electric railway engineers and operating people have been made to realize that the overhead-contact system is as important as, if not more important than, any detail of the system. This is on account of the fact that with locomotives and substations it is possible to hold units in reserve for emergency use, whereas with the overhead-contact system there is no such thing as reserve capacity. Furthermore, the contact system, if properly designed and built, is a permanent way equal in importance to the track and road bed. With the exception of using wood poles for supports, on account of the plentiful local supply, it is believed that the B., A. & P. railway is about as near permanency in its overhead construction as can be expected at the present stage of the art. The authorities differ as to the use of a rigid type of construction as compared with one which is flexible, though the majority agree that until we can construct a perfect track and road bed the flexible overhead-contact system is far superior to the rigid type. The engineers of the B., A. & P. railway adopted the flexible construction, which is of the catenary type, with 0.5-in. Seimens-Martin galvanized strand for messenger and 4/0 grooved copper trolley wire for contact member, the trolley wire being hung from the messenger by specially designed loop hangers which permit absolute freedom in vertical movement of trolley wire when subjected to pressure imposed by pantograph trolley. It is believed that on the main line, where operating conditions require relatively high speed, they have the only contact wire in this country which can be truly classified as flexible, and which is practically free from changes in flexibility, with consequent damage to contact wire from current-collecting devices.

The various yards and sidings included in the electrified section contain 300 track switches, calling for a corresponding number of overhead special work devices. Heretofore it has been customary to make use of grids or other devices, known as deflectors, for insuring the safe transition of the current collector from main line to siding or the reverse. On this work these devices, which are expensive to install and maintain, have been dispensed with and such transition is

taken care of by holding the contact wires parallel and in the same plane for a certain distance and raising the turn-out wire at the end so as to give a gradual approach to level of main line wire and current collector.

One of the great problems in connection with the overhead-contact system has been to take care of the expansion and contraction of various members due to changes in temperature and the fact that copper and steel, with their different temperature coefficients, make up the main members of the structure.   The B., A. & P. Co. are now experimenting with a system of counterweights for maintaining uniform tension in contact wires regardless of temperature changes and they promise to prove entirely satisfactory.

Though as yet only a portion of the entire equipment is in service, those concerned feel justified in believing that the performance in general of the electrical equipment, including locomotives, substations, and overhead, is nothing less than remarkable, in view of the fact that practically every detail is all new special design and that the operation of the equipment has been in the hands of individuals who did not have the advantage of previous training to fit them for the work, and they feel assured that when the entire service is taken over the results will be equally as satisfactory.

The Compressed Air System of the Anaconda Copper Mining Co., Butte, Mont.

BY BRUNO V. NORDBERG, MILWAUKEE, WIS.

THE high cost of coal in Butte and the development of large amounts of cheap electric power from the Missouri river caused the Anaconda Copper Mining Co. in 1908 to make an investigation as to the possibility of adapt-
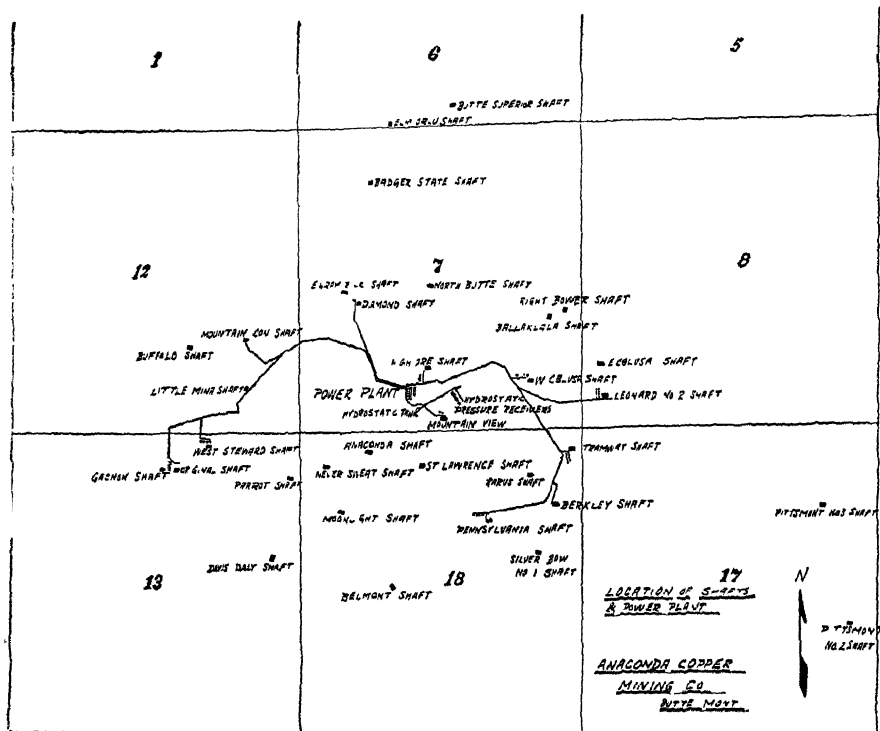
FIG. 1.—MAP OF BUTTE MINING DISTRICT.

ing this cheap electric power for the operation of their many large hoisting plants. At this time electric power had already been adopted for practically all other power purposes around the mines, such as pumping, compressing air, etc.

The map (Fig. 1) shows the location of the mines of the Butte mining district. These mines are located within a circle of approximately a 1-mile

radius.   The shafts are all vertical, the maximum depth being 2,000 to 2,800 ft. at present, but sinking to greater depth is in progress in several shafts which are expected, in the future, to reach an ultimate depth of at least 3,500 ft.   The hoisting is done from a great many levels between 1,000 ft. and bottom of the mine.

Every mine has its own dry house, which requires a boiler plant.

For the purpose of making this investigation it was determined to make extensive tests of four representative steam hoists of the district, namely, at the Speculator shaft of the North Butte Mining Co., and the Diamond, High Ore and Mountain View shafts of the Anaconda company.

At the Speculator shaft the hoist is a Nordberg, having steam cylinders 32 and 32 by 72 in. with Corliss valve gear.   The drums are smooth, 12 ft. in diameter, carrying 1.5-in. round rope in two layers, operated by means of clutches.

At the Diamond shaft the hoist was originally built by the Risdon Iron Works but had been twice rebuilt by the Anaconda Copper Mining Co. It had steam cylinders 30 and 30 by 72 in. with Corliss valve gear and link motion reverse.   It was equipped with reels carrying flat rope 0.5 by 7 in.

At the High Ore shaft, the hoist was built by the Union Iron Works. The cylinders were 30 and 30 by 72 in. with poppet valves operated by wrist plate motion with link reverse.   This hoist was also equipped with reels carrying flat rope 0.5 by 7 in.

The Mountain View hoist was a Webster, Camp & Lane with cylinders 28 and 28 by 72 in., having Corliss valves with link motion reverse.   It was equipped with reels carrying flat rope 0.5 by 7 in.

In passing it should be noted that several of the hoisting engines at Butte have to be located very close to the shafts, which necessitates reels and flat ropes to eliminate an otherwise prohibitive fleeting angle.

The tests were made by H. W. Dow and B. V. Nordberg, Jr., representatives of the Nordberg Manufacturing Co.   Those directing the tests deserve great credit for the persistency with which they worked to get a sufficient number of readings to give an accurate record of the hoistings, without which it would not have been possible to determine the different factors entering into the power problem of the hoisting system.

The main object of these tests was to determine the power required and the nature of the load.   In order to do this it was necessary to determine the exact time and duration of every trip, the load hoisted and the depth from which it was hoisted in or out of balance, as well as the speed conditions during each trip.

In order to get a record of the hoisting operations, a Karlik tachograph was used.   This instrument draws a velocity diagram of every trip made

by the hoist with the velocity as ordinates and hoisting time as abscissæ, and gives a 24-hr. record on a paper 8 by 42 in.

The areas of the different diagrams represent the hoisting depth, but as these diagrams are very small and do not give the weight hoisted, it was necessary to obtain written logs of the hoisting for periods extending over 24 hours. Several such logs were obtained in which were noted the exact time of starting and stopping for each trip, the nature of the load hoisted or lowered (ore, waste, men, tools, etc.), the depth from which it was hoisted, or the distance lowered, and a statement showing if the hoist was in or out of balance. In order to prepare such a log the investigators had to stand watches of 24 hr. and carefully record every movement of the hoist during that time. This they did, so that the tachograph cards, which were a check on their vigilance, absolutely correspond with the written logs.

In order to determine accurately the weight of the rock handled per skip load, the ore bins at the North Butte mine were thoroughly cleaned just before and just after writing a 24-hr. log. All of the ore hoisted during this period was loaded into marked cars and the gross tare and net weights obtained from the smelter when unloaded. To determine the weight of the men it was found, by repeated trials that eight men, or the load per cage, as they would ordinarily fall into line, would average 160 lb. each.

All the odd material, such as timbers, supplies and poor rock handled up or down the shaft, was weighed at the surface. With all these weights, supplementing the written log, the shaft horsepower was calculated for each of the mines over a 24-hr. period.

Continuous indicator cards were taken, hoisting from every level, and hoisting every different kind of a load in balance and out of balance, and note was made on the tachograph card of the trips during which the indicator cards were taken.

Crosby continuous indicators were used and provided with electric pencil attachment, so that all four indicators worked simultaneously. By this means cards were obtained for every revolution made by the hoist for every given trip, thus giving a perfect record of the mean effective pressure and the resultant indicated horsepower.

In order to determine accurately the instantaneous revolutions per minute of the engine or rope speed corresponding to each individual indicator card, a special instrument was constructed. This instrument carried a continuous sheet of paper 15 in. wide which traveled at a perfectly uniform rate of speed, thus measuring time accurately. Moving at right angles to the travel of the paper was a pen having a reduced motion taken from the shaft driving the hoist miniature. Thus the curves drawn by this instrument had time as abscissæ and revolutions

of the engine or rope travel as ordinates. In what follows these curves will be referred to as "time distance" curves, or diagrams.

These curves were constructed on a sufficiently large scale so that velocities at any point can be determined by finding the value of the tangent to the curve at that point, and after having constructed such a velocity diagram, acceleration at any point would be determined by finding the value of the tangent to the velocity curve at that point.

In this paper, the term shaft horsepower means the actual net unbalanced load in pounds multiplied by the distance hoisted or lowered in feet, reduced to horsepower and does not include friction of any sort.

The following figures give a fair idea of the conditions under which the large hoisting plants in Butte have to operate:

Balanced Hoisting.

Ore hoisted per trip, 5 to 6 tons.
Hoisting depth average, 1,800 ft.
Hoisting ore in balance, 31 loads in 35 min. from 1,800 ft. depth, equivalent to 582 average shaft horsepower for full load of 6 tons.
Duration of single trip, 51 sec. average.
Work to be done per single trip, 770 shaft horsepower.
The following are the particulars of the loads carried on these hoists:

	Kind of Hoist	Weight Rope for 1,800 ft. depth, lb.	Skip and Cage, lb.	Total Dead Load, lb.	Normal Load Ore, lb.	Ratio Dead Load to Ore Hoisted
Speculator....	Drum	4,800	11,000	15,800	10,000	1.58
Diamond.....	Reel	10,700	11,000	22,700	10,000	2.27
High Ore.....	Reel	10,700	11,000	22,700	8,000	2.84
Mountain View	Reel	10,700	11,000	22,700	10,000	2.27

The "dead loads" are here given for the average hoisting depth which is 1,800 ft. It will be noticed that these dead loads are much greater than in average hoisting practice. This is largely due to the Butte practice of permanently attaching a cage to the skip, the combined weight of which is equal or greater than the weight of ore hoisted, while in average hoisting practice the skip weighs from 60 to 70 per cent. of the ore it contains. Then again, when flat ropes are used to go down to 3,000-3,500 ft. depths, these ropes become very heavy and increase the dead loads correspondingly.

In average hoisting practice the dead loads for 10,000 lb. of ore hoisted per trip would be 10,400 lb. if the hoist was carrying strong enough rope for an ultimate depth of 3,500 ft.

The magnitude of the dead loads in a hoist is an important factor in the mechanical efficiency of the hoisting operation, as it influences the friction and the dynamic energy that has to be supplied to the hoist in getting up to speed and released when retarding the hoist. Ordinarily this dynamic energy is lost in form of heat on the brake shoes.

Occasionally it is required to lift unbalanced loads from various depths. In some mines such unbalanced loads come most frequently from the bottom of the mine. At North Butte considerable unbalanced hoisting was done from 2,200 ft. depth. Hoisting the skip and cages weighing 11,000 lb., one-half of the rope, 4,000 lb., and the load of 12,000 lb., or a total of 27,000 lb. from 2,200 ft. gives us:

$$\frac{27000 \times 2200}{33000} = 1800 \text{ shaft horsepower}$$

the work being performed in one minute.

At the Mountain View some of these unbalanced loads represented about 2,000 h. p. at the shaft. The average work per day of these hoists ranges from 25 to 110 shaft horsepower, based on the productive work.

The chart (Fig. 2), in which the effective work performed at the shaft is plotted over the dial of a clock, gives a good idea of the continuity of hoisting at these mines. The charts show the work done at the Speculator Shaft Jan. 11 and 12, 1909, and is a good average of the operation at the mine. Only the productive work, that is, the work represented by the hoisting of ore, is plotted.

From the tachograph diagrams and the log, the weight hoisted and the depth were determined, also the time during which the work was done. Thus, for instance, there were hoisted from 12:00 to 12:45 a. m. 31 loads from 1,800 ft. depth, or 155 tons, representing an average work of:

$$\frac{155 \times 1800}{45} = 6200 \text{ foot tons per minute} = 376 \text{ shaft horsepower during that period.}$$

The variation in the load during the period under consideration as determined from continuous indicator diagrams (Fig. 3), time distance curves (Fig. 4) and tachograph cards (Fig. 5) is shown in Fig. 6.

Frequently unbalanced hoisting was done when one compartment was obstructed. Figs. 7, 8 and 9 show tachograph card, power curve and continuous indicator diagrams taken March 8, 1909. Following a period of hoisting ore in balance from the 1,800 ft. level the station tenders were lowered to the 2,200 ft. level and ore hoisted from there out of balance.

A glance at the diagrams showing the power when hoisting in balance from 1,800 ft. depth (Fig. 6) shows that the horsepower fluctuates from +2,300 to — 1,600 i. h. p., the average during this period being 630 i. h. p.
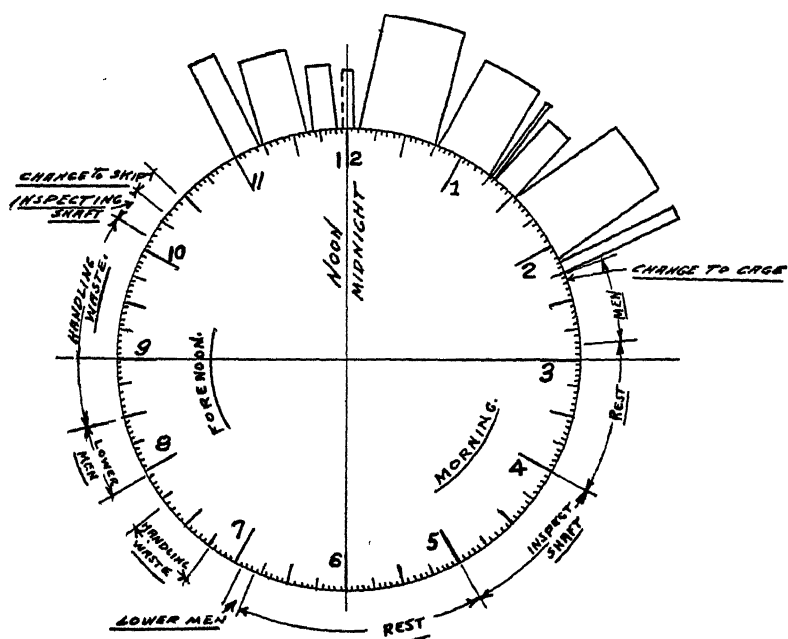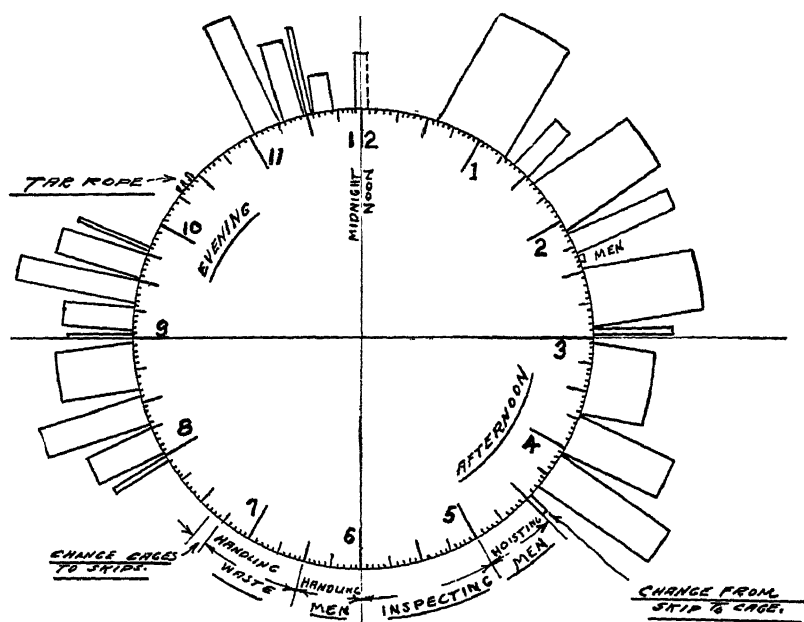
FIG. 2.—DIAGRAM SHOWING PRODUCTIVE WORK AT SPECULATOR SHAFT, NORTH BUTTE MINE.
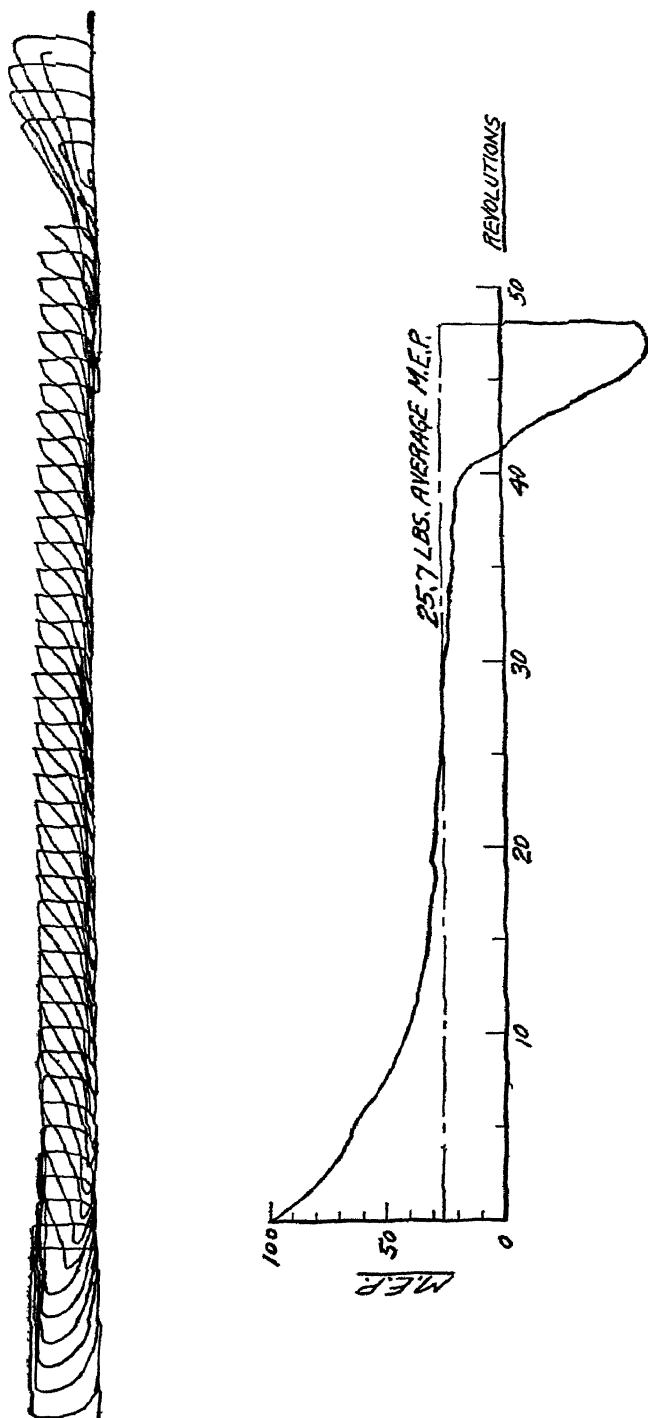
FIG. 3.—CHARACTERISTIC CARD WHEN HOISTING WITH A 32-32 BY 72 STEAM HOIST AT NORTH BUTTE MINE. BALANCED LOAD, 10,280 LB. OF ORE; LIFT, 1,800 FT.

REV. 48 45 | 40 | 35 | 30 | 25 | 20 | 15 | 10 | 5

SECONDS 50 45 40 35 30 25 20 15 10 5
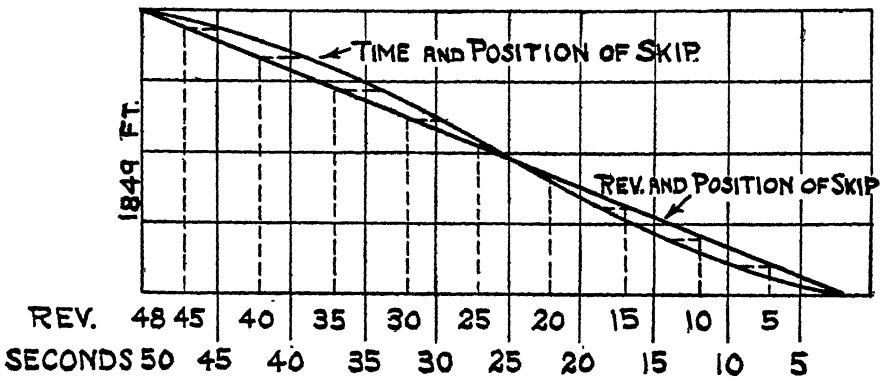
FIG. 4.—TIME DISTANCE CURVE.

A.M. 12:10 :11 :12 :13 :14
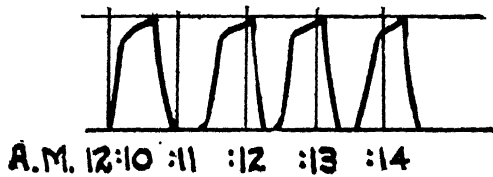
FIG. 5.—SECTION OF TACHOGRAPH CARD.
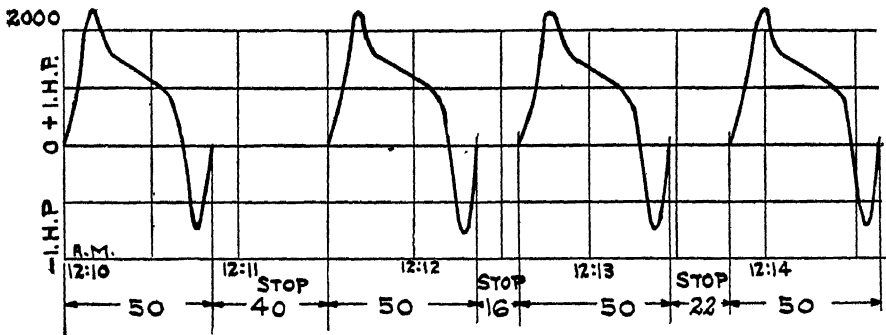
FIG. 6.—BALANCED HOISTING FROM 18TH LEVEL, SPECULATOR SHAFT, NORTH BUTTE
MINE, JAN. 12, 1909, 12 : 10 TO 12 : 14 : 40 A. M.

Average indicated horsepower per trip = 880.
Maximum indicated horsepower = 2,380.
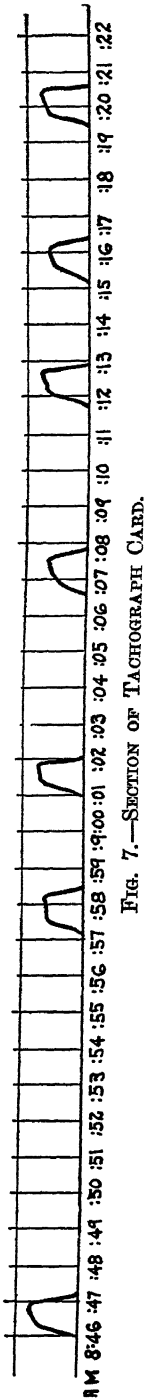
Fig. 7.—Section of Tachograph Card.
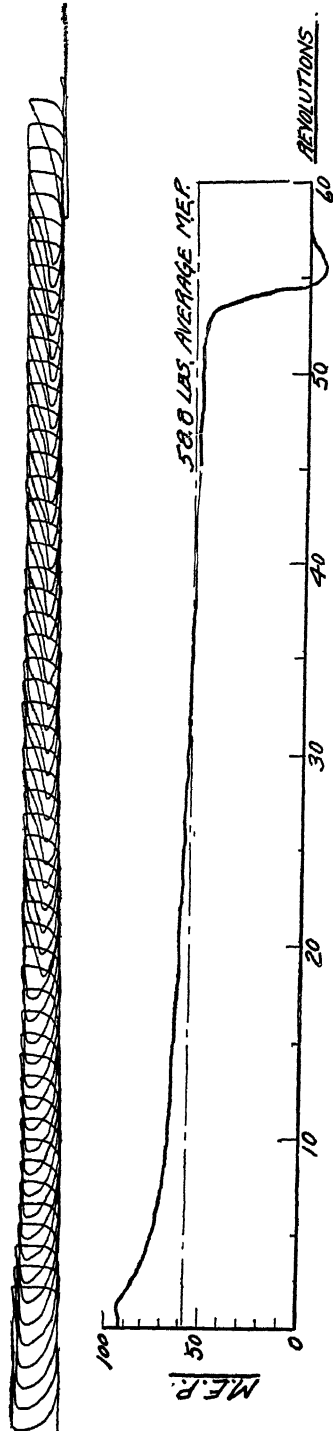
Fig. 9.—Characteristic Card when Hoisting with a 32-32 by 72 Steam Hoist at North Butte Mine. Unbalanced Load, 25,215 lb. of Ore; Lift, 2,200 ft.
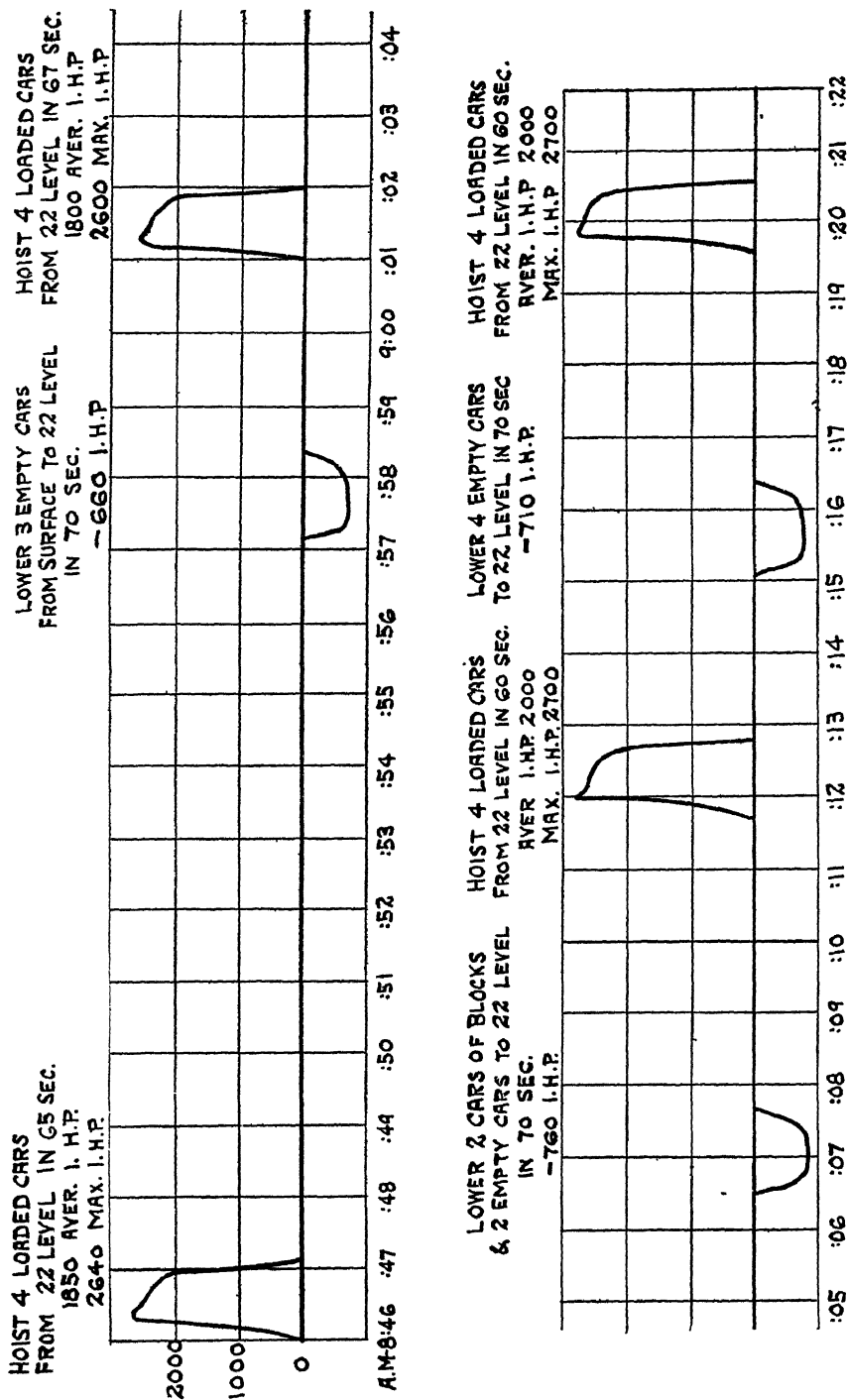
FIG. 8.—UNBALANCED HOISTING FROM 22D LEVEL, SPECULATOR SHAFT, NORTH BUTTE MINE, MAR. 8, 1909, 8 : 46 TO 9 : 21 A. M

This represents very nearly the average balanced hoisting condition at these mines.

From the diagrams of unbalanced hoisting from 2,200 ft. depth, Fig. 8 shows that during 34.5 min. the horsepower fluctuates from + 2.700 to − 900 h. p., the average during this period being 160 h. p. It has already been pointed out that most of the unproductive work is done out of balance. Occasionally a load averaging 2,000 i. h. p., with a peak 2,700 i. h. p. follows right after one averaging 900 i. h. p.

Much energy was also used in hoisting waste from lower to upper levels out of balance. In fact, it was found that although in all the large producers of the Butte mines an auxiliary hoist was installed to relieve the main hoist of non-productive work, still the non-productive and unbalanced hoisting performed with the main hoists covered fully as much, or more energy, than the productive work. That is largely due to the fact that most of the unproductive hoisting is done out of balance and that the "dead loads," i. e., ropes, skips and cages, are usually heavy. Thus, the skip and cage attached thereto at the Speculator shaft weighed about 11,000 lb. while the load in the skip was 10,000 lb.

The following table gives a good idea of the continuity of hoisting and the ratio of productive to non-productive work. The work done is expressed in effective horsepower: For 24-hr. run,

$$\text{Effective horsepower} = \frac{\text{Total W't hoisted (lb.) x distance in ft. hoisted}}{\text{Total actual time (min.) hoist is in motion x 33,000}}$$

Effective Horsepower.

	Productive or Hoisting Ore	Non-Productive			Ratio Prod. Work to Non-Prod. Work	Percentage of time during which hoist is doing positive work
		Men	Material	Total Non-Productive		
Speculator.....	565	433	758	1,191	1:2.12	24 0
Diamond......	706	449	630	1,079	1:1.52	19.6
High Ore	456	713	813	1,526	1:3.34	12.5
Mountain View.	508	493	631	1,124	1:2.22	21.6

The power diagrams Figs. 6 and 8 are derived from the continuous indicator cards by measuring up every individual stroke diagram with a planimeter, determining the mean effective pressure, and, by aid of the time-depth diagram, ascertaining the time necessary to perform each revolution of the engine. From these quantities the indicated work expressed in horsepower was calculated for each revolution of the hoist and

plotted on time basis in the power diagrams. The diagram (Fig. 6) for balanced hoisting shows during the retardation periods a negative performance, i. e., the engine pistons during these periods did work upon the steam. This is the result of the practice in vogue of "plugging" the engines. To do this the valve motion is reversed at the commencement of the retardation period and the throttles kept partly open. The bypass is also kept partly open. The effect is then to retard the motion of the pistons with steam from the boilers, which steam escapes to the opposite side of the pistons and from there to the atmosphere as the exhaust valve is open on that side. The quantity of steam thus escaping and its pressure is regulated with the bypass valve. The continuous indicator cards show the retarding effect due to this mode of control, which results in the development of a negative work of nearly 1,000 h. p. during part of the retardation period. It is plain that the steam used when "plugging" is absolutely wasted, but as will be shown later on, an improved method of "plugging" was devised when these hoists were remodeled, by which the greater part of the energy used in retarding the hoists is returned to the power system.

It was found that the mechanical efficiency of these hoists varied greatly and that the friction was principally in the shaft guides and head sheaves. This was proved by the fact that in all cases the percentage of friction is much less when hoisting out of balance with one drum or reel held by the brake than when hoisting in balance. Thus we found that when hoisting from a depth of 2,200 ft., the friction of the hoisting engine, ropes, sheaves and cages, including the windage, was as follows:

	In Balance. Per Cent.	Out of Balance. Per Cent.
Speculator	21.1	13.0
High Ore	23.0	17.5
Diamond	29.0	10.0

The friction was determined by hoisting a weighed load and taking continuous indicator diagrams while the load was hoisted. The friction is then the difference between the indicated work and that represented by the load hoisted 2,200 ft. The percentages given refer to the indicated work. There can be no doubt that the high values of friction found in some of the shafts are due to defects in the shafts and it is very probable that in some of the cases one compartment was more defective than the other. In the Diamond shaft, for instance, the compartment in which the unbalanced load was hoisted had free guides, while the cage did not run freely in the other one when the load was hoisted in balance. It is to be regretted that we did not in all cases test out the friction of both compartments of the shaft, in which case we would have been able to present more complete data on this subject.

In one of the shafts we found as high a friction as 40 per cent. of the indicated work. This shaft was, however, in bad shape at the time. All friction tests conducted on unbalanced hoisting gave consistent results, the lowest friction being obtained when hoisting at a low speed. The friction seemed to increase very considerably with the speed, which indicates that the effect of windage is greater than has been generally assumed. There are very few data available on friction of a mine hoist. Had we suspected, at the time we made our tests, what the results were to be we would have investigated the subject more fully. It is plain,
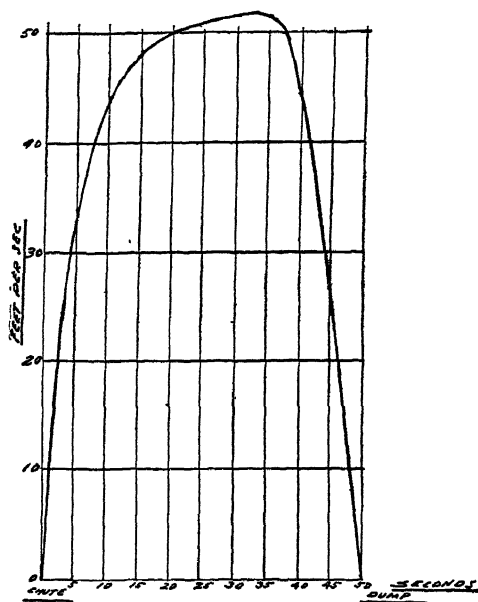
FIG. 10.—VELOCITY CURVE. BALANCED HOISTING FROM THE 18TH LEVEL OF THE SPEC-
ULATOR SHAFT, NORTH BUTTE MINE.

from the results obtained when hoisting out of balance, that the friction of the engine proper is a very small quantity, probably less than 6 per cent. in any of the engines tested.

Fig. 10 shows a rope speed curve when hoisting in balance from the 1,800 ft. level with a drum hoist. The load is hoisted in 50 sec. at an average rate of about 36 ft. per second, the maximum speed reaching about 52 ft. per second. The initial acceleration is about 8 ft. per second and the retardation about 5.6 ft. per second.

Fig. 11 shows the rope speed curve of the reel hoist at Diamond mine lifting the load in balance from the 1,800 ft. level. The load is here hoisted in 48 sec., so that the average speed is practically the same as in case of the drum hoist, but as the acceleration and retardation here average

only about 2.5 ft. per second, the maximum speed reaches as high a value as 72 ft. per second. The rope speed curve is here made up of the acceleration and retardation curves, meeting at the point of maximum velocity. This seems to be the characteristic of most flat rope hoists.

The foregoing shows that the hoisting is periodical in all these mines
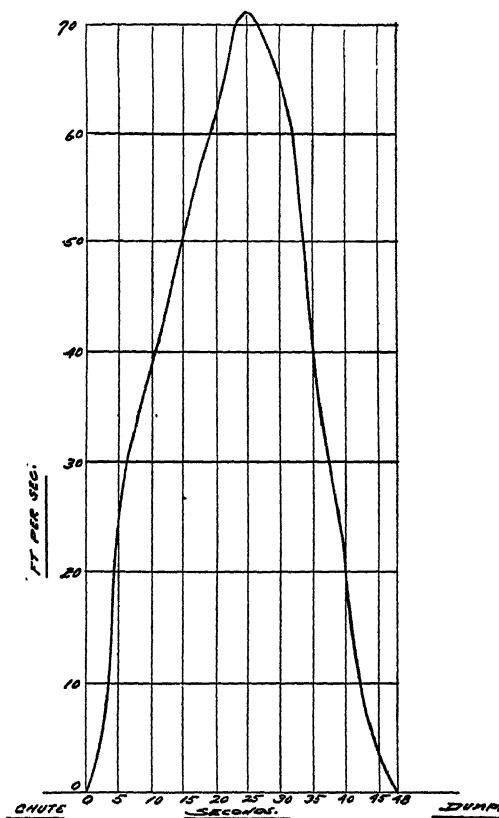
FIG. 11.—VELOCITY CURVE. BALANCED HOISTING FROM THE 1,800-FT. LEVEL OF THE DIAMOND MINE; 30–30 BY 72 REEL HOIST.

and done at a high rate of speed with high acceleration of the loads, the masses to be accelerated being excessively large.

The average of the work performed by the auxiliary hoists was in most cases found to be negative, i. e., the work done in lowering material into the mine exceeded that done in hoisting out of the mine. Thus, the positive work done in hoisting with the auxiliary at the Speculator mine was found to be 995,000,000 ft.-lb., while that done in lowering men, timber, tools, etc., was 1,073,000,000 ft.-lb. per twenty-four hours, making a total of minus 78,000,000 foot pounds per twenty-four hours.

In several mines, where large quantities of waste rock are lowered from

the surface down into the mine, the negative work performed is of such magnitude that if it could be recovered and the energy stored, a considerable saving would be the result. We shall show further on how this was accomplished.

Accurate records are available showing the work done and energy used by the steam hoists at the largest producing mines at Butte. It would take up too much space to exhibit these records in their complete form. The examples given above will, however, give a good idea of the magnitude and nature of the load to be handled by the Butte hoists.

When, in 1909, it was decided to operate these hoists by power, transmitted electrically to Butte, the question was discussed as to how much these existing hoisting conditions could be modified and improved. Any system of dispatching whereby the starting of two or more hoists simultaneously could be prevented was objectionable from a mining point of view. The periods of heavy hoisting, with corresponding periods of light or no hoisting, as illustrated by Fig. 2, could not be materially changed without seriously interfering with the underground operations.

It was then planned to regulate the hoisting so that these heavy periods would not coincide at all the mines as they did at that time. Up to the present time, however, this condition has not been changed materially.

Furthermore, the mining company desired the hoists not only to operate under the conditions already outlined for the steam hoists, but to possess considerable flexibility to allow for greater hoisting depths in the future and also for greater rates of production. After consideration of all these facts the Anaconda Copper Mining Company decided not to attempt to employ any method of operating the hoist by the application of electricity directly to them but to adopt the compressed air system of hoisting designed by the writer.

The power system for operating the hoists of the Anaconda Copper Mining Company mines comprises:

(a) An electrically operated air compressing plant in which air is compressed to 90-lb. pressure.

(b) The electric current is generated by water power located at a distance of 150 miles from Butte. In order that a maximum amount of energy can be transmitted through such long transmission line it is important that the work is perfectly equalized and all peaks in the load eliminated. This requires an air storage of great capacity and one in which the stored energy can be held without loss for extended periods.

(c) Air reheating plants at the different shafts.

(d) The hoisting engines were provided with new cylinders designed for economical use of the compressed air. The old and wasteful auxiliary hoists were replaced with new ones, specially designed for compressed air.

The air storage plant was connected with the rock drill system so that

during the idle periods when little or no air was used by the hoists, the air could be turned into the rock drill system.

At present upward of 25 per cent. of the total capacity of the compressor station is sent into the mine. This not only helps out the mine system but provides a high load factor for the compressor station.

Description of the Plant.

*The 100,000-volt substation at Butte is located near the center of the district in which the power is distributed. The substation building is 150 ft. by 50 ft. in plan and 50 ft. high. It is a brick-walled, steel-framed structure with concrete floors and roof.

There are installed at present two banks of single-phase transformers, rated at 3,600 kw. per bank and two banks rated at 7,200 kw. per bank, connected in delta on both high and low tension sides. They step the voltage down from 102,000 to 2,500, at which voltage it is distributed to customers. The transformers are installed in fireproof compartments, entirely shut off from the rest of the building by brick walls and opening only out of doors. The transformers are mounted on wheels and can readily be run out on to a flatcar which stands on a track running parallel with the building in front of the row of transformer compartments. This arrangement furnished a convenient method of handling the transformers, both at the time of installation and afterwards, in case it is necessary to make repairs.

On the gallery above the transformer compartments are located the electrolytic lightning arresters. On the gallery opposite are the 100,000-volt line switches. Possibly the most unique feature of the electrical layout is the 100,000-volt bus construction. For flexibility in switching, duplicate busses are provided. The busses themselves are made of 1.5-in. iron pipe suspended by standard line insulators from the roof trusses of the building. The three conductors of each three-phase bus are suspended one above another, each being supported by the next one above. The connections to, the lines are also of iron pipe, making the bus structure as a whole quite rigid and well adapted to the use of suspension insulators.

The switchboard is in two sections, one section operating all line and transformer switches, which are remote controlled; the other section taking care of the 2,500-volt feeders, which are controlled by hand-operated automatic switches.

All electrical apparatus in the substation was supplied by the General Electric Co.

The load supplied in Butte is confined entirely to the mines, the power being used chiefly for the operation of motor-driven air compressors and electrically driven pumps.

*Substation—reprinted from an article by Max Hebgen in the "General Electric Review."

The load is very nearly uniform for twenty-four hours each day through-out the year. The load factor is, in fact, close to 90 per cent.

Description of Compressor Station.

In the compressor house which is located close to the substation there are installed six Nordberg two-stage compressors, three of which are fitted with variable capacity valve gear for automatically varying the capacity, while the other three run at a fixed capacity. The compressors are directly connected to Westinghouse synchronous motors and each compressor runs at a speed of 76 rev. per min. Fig. 12 shows an interior view of the compressor house. The cylinder dimensions of all compressors are 30 and 50 by 48 in. The pistons of these compressors are supported outside the cylinders, on crossheads running in oiled guides, the piston rods being very large in diameter so as to insure minimum deflection. Only the packing rings touch the cylinder walls. The variable capacity compressors have Corliss inlet valves and automatic discharge valves, the capacity being regulated by the closure of the inlet valves at different points of the stroke by means of a releasing mechanism under control of a pressure regulator. The constant capacity compressors are fitted with positively operated Corliss valves for inlet and discharge. These compressors show a high efficiency. A set of indicator cards from these compressors is shown in Fig. 13. The capacity of the compressors is at 76 rev. per min.—7,650 cu. ft. of free air per minute. The atmospheric pressure at the power plant is 12 lb. absolute and the normal air pressure 90 lb. gauge, or the same as used on the rock drill system, but when cards (Fig. 13) were taken the discharge pressure was 98.2. The indicated work is 1,099 h. p., while the theoretical work of perfect two-stage compression with perfect intercooling and no pressure losses is 1,045 h. p. referred to the actual air compressed.

As the starting of these compressors, if done by the electric motor in the ordinary way, would throw peaks of considerable magnitude upon the electric system, a new method of starting was adopted whereby such peaks are entirely avoided. The valve motion of the compressors was fitted with a reversing gear whereby the valves, for the purpose of start-ing, are so adjusted as to turn the machine into an air motor. In the Butte power system there is always a large volume of air stored and main-tained under 90 lb. pressure. The compressors are thus started and brought up to speed by the energy in the stored air. When up to speed and in synchronism with the generators 150 miles away and with other compress-ors running, the electric current is switched on and valve motion reversed. The machine immediately starts to compress air. Thus, when starting no electricity is used and when the air compression begins with the com-
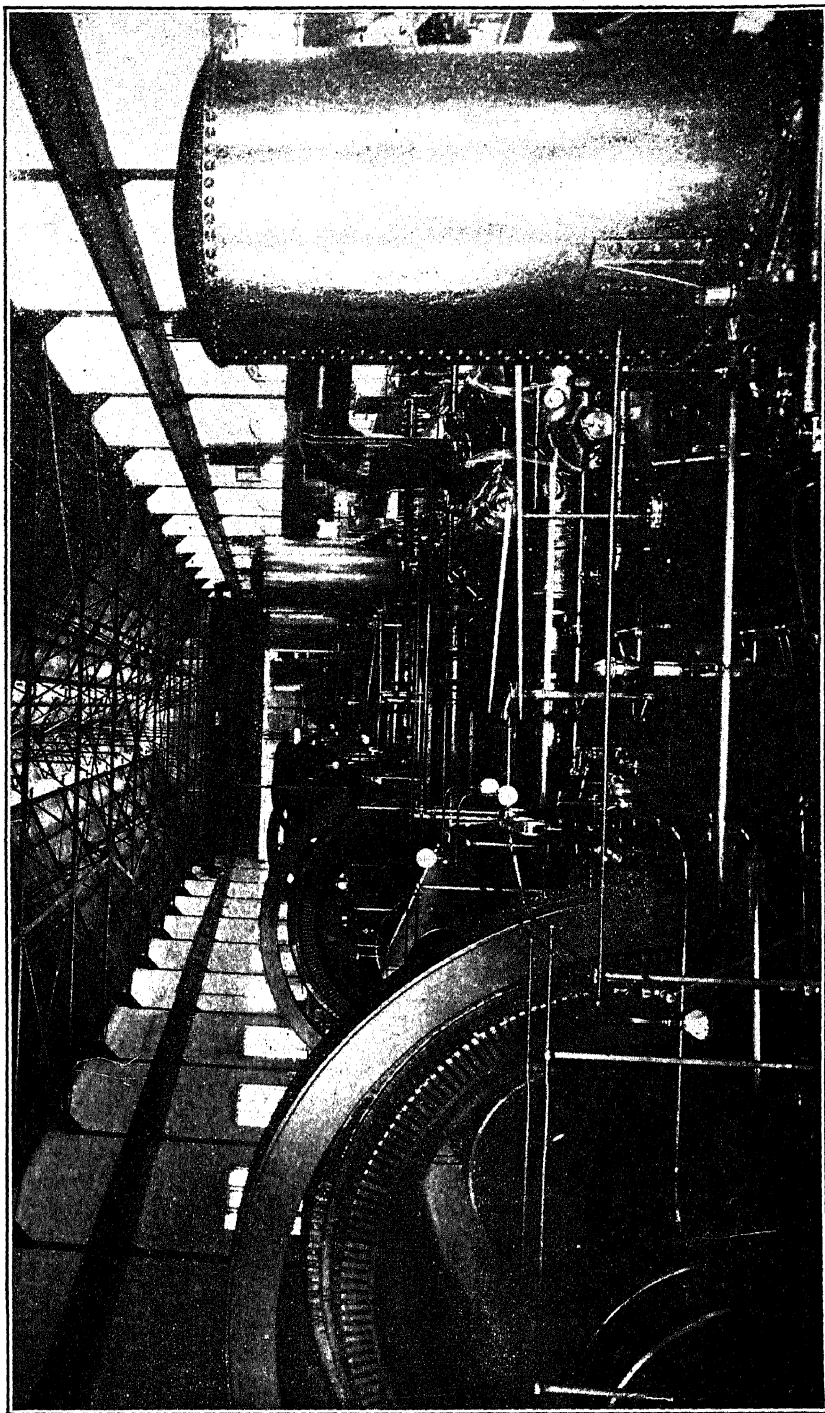
FIG. 12.—INTERIOR OF COMPRESSOR PLANT.

pressor up to speed and in synchronism, the power is only the normal power necessary for the compression. This starting gear works very satisfactorily. At the end of April and beginning of May, 1912, Butte was visited by several severe electric storms which caused interruption in

FIG. 13.—DEVELOPED AIR CARDS. CONSTANT CAPACITY COMPRESSOR

the operations of the compressors several times. At that time there were four compressors installed, three running and one in reserve. On May 2, 1912, during a particularly severe storm, all three compressors were stopped by lightning. The shutdown was not of long enough duration to interrupt the hoisting, and the time required to start the three compressors was less than four minutes from the moment when the first

FIG. 14.—VIEW OF VARIABLE CAPACITY COMPRESSOR.
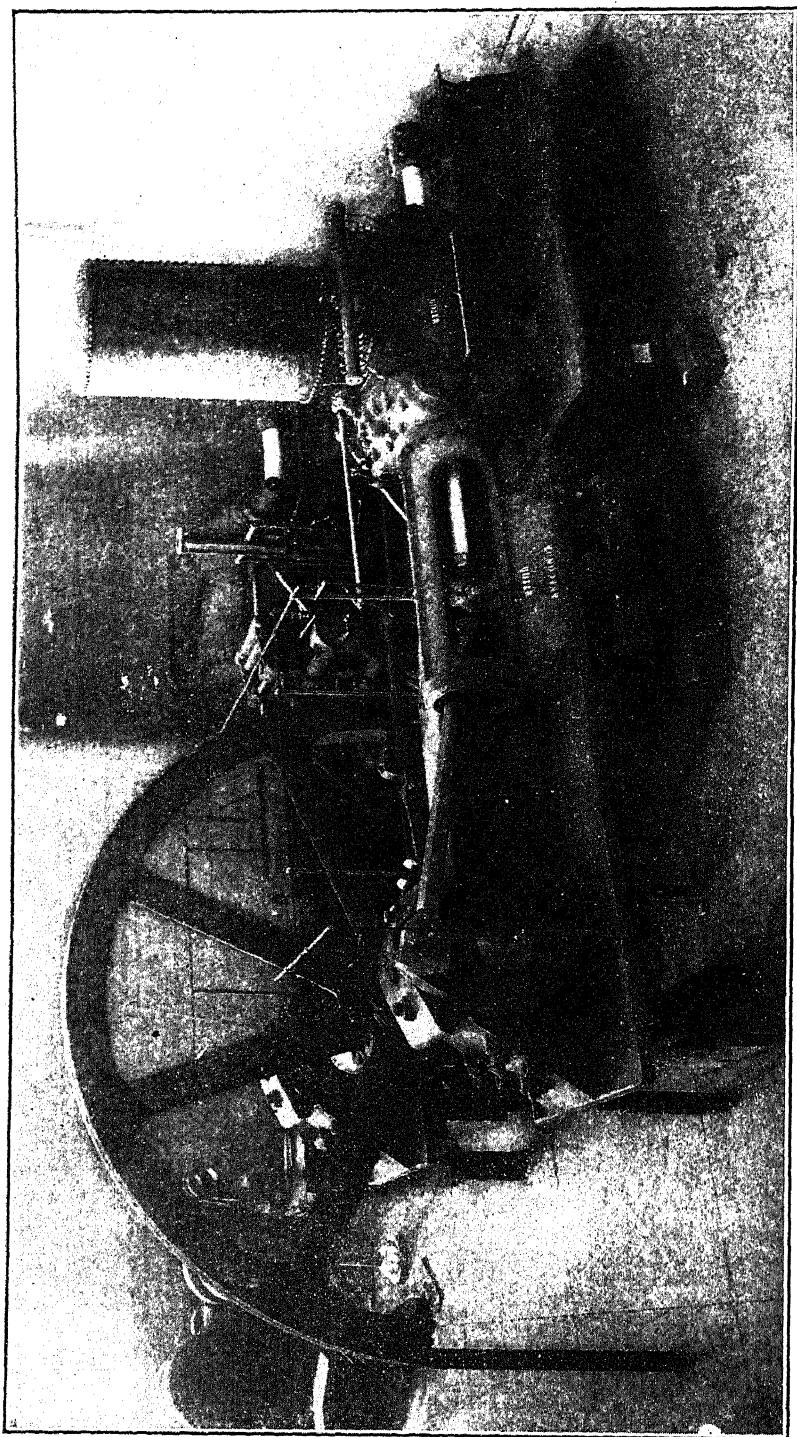
FIG. 15.—VIEW OF CONSTANT CAPACITY COMPRESSOR.

compressor started until all three were in step, reversed and compressing air into the system.

The compressors are connected to a number of large air receivers and these to the distributing main in a manner as shown in Fig. 17, from which can be seen that any of the receivers can be shut off from the system for inspection or repairs, there being a valve on each side of the receiver for that purpose. Figs. 14 and 15 show respectively the variable and the constant capacity compressors.

Air Storage.

In order to equalize the load on the compressor station, it was necessary to provide for two kinds of peak loads on the hoists, first—the peak due to the acceleration of any hoist. These peaks are very large but last for a few seconds only. To take care of this condition, storage receivers of sufficient capacity to liberate sufficient air for the acceleration of the hoist without excessive pressure drop were installed at each hoist. The highest rate of air consumption at the large hoists is at the rate of about 60 cu. ft. of compressed air per second for 5 sec. This condition determines the air receiver capacity necessary, each requiring four receivers of 2,100 cu. ft. volume. It will be readily understood that as the pipe lines from the compressors to the different hoists on the system are of considerable length that the capacity of the pipe lines are of little value as storage reservoirs unless accompanied by a prohibitive pressure drop. The installation of the above receivers therefore allows the pipe lines to be designed for average flow.

The amount of energy that can be stored in the receivers or expansion tanks is, however, limited. Calling the volume of compressed air supplied to each receiver during any given length of time v, the volume drawn from the receiver by the hoist during the same time v_1, and the volume of the receiver V gives us ·

$$V = \frac{v_1 - v}{\dfrac{1 - 1}{r}} \text{ and } r = \frac{V}{V + (v_1 - v)}$$

if r is the ratio of the final to the initial pressure in the receiver during the same time.

During the acceleration period when the maximum rate of air consumption may reach 60 cu. ft. per second for 5 sec. $5 \times 60 = 300$ cu. ft. for 5 sec., we get, if air is supplied to the receiver at the rate of 15 ft. per second, under which condition there is no appreciable drop of pressure in the pipe line:

$V = 4 \times 2100 = 8400$ cu. ft. $v = 5 \times 15 = 75$ cu. ft. $v_1 = 300$ cu. ft.

$$v_1 - v = 275.$$

$$r = 8400 \frac{8400}{8400 + 275} = \frac{8400}{8675} = 0{:}97$$

so that, if the initial pressure in receiver is $90 + 12 = 102$ lb. absolute, the final pressure will be $102 \times 0.97 = 99$ lb., or the pressure drop 3 lb. As the air in the pipe line is accelerated during the acceleration period of the hoist, and therefore attains a higher rate of speed than 15 ft. per second, the pressure drop during the period will actually be less than 3 lb. The pipe lines are made of such diameter that they will allow a flow of 2,700 cu. ft. of compressed air per minute or 45 cu. ft. per second with a pressure drop of 5 lb. The heaviest unbalanced loads call for an air consumption of 40 cu. ft. per second and as they are hoisted in one minute of time, 2,400 cu. ft. of compressed air will during that time flow into the hoist cylinders.

The pressure drop will, therefore, be less than 5 lb. as less air is taken out of the receiver than can be supplied to it from the pipe line with 5 lb. drop.

Each of the large hoists is fitted with four storage receivers aggregating 8,400 cu. ft. capacity and there are similar receivers installed at the compressor plant. As all these receivers are connected by the pipe lines they will act together in case of any heavy demand of air, no matter if this air is demanded at one hoist or at several hoists simultaneously. In order to show in how far remote these receivers can equalize the demand for power if heavy hoisting is going on simultaneously in several mines, we will assume that in three of the largest producers hoisting is in progress from 1,800 ft. depths and that in each 36 loads are raised in 36 min. This is a condition that actually exists and must be met, and each trip requires 1,012 cu. ft. compressed air. Three compressors are running, each supplying 880 cu. ft. compressed air per minute, or 2,640 cu. ft. per minute for the three.

There are at the different mines connected to the system 40 air receivers and on the compressors 12 receivers, or a total of 52 receivers, each of 2,100 cu. ft., so that the aggregate receiver volume will be $52 \times 2,100 = 109,200$ cu. ft. We have thus:

$$v = 2640 \times 36 = 95040$$
$$v_1 \ 1012 \times 3 \times 36 = 109296$$
$$V = 109200$$
$$r = \frac{V}{V + (v_1 - v)}$$
$$v_1 \ v - = 14256$$
$$r = \frac{109200}{123456} = 0.886$$

At the end of the 36-min. period there is thus a pressure on the system of 102×0.886 = 90 lb. absolute. The air pressure has then dropped 12 lb. during that time. From the above can be seen that air receivers placed in the pipe line are not very effective when it is required to store much energy. Such receivers can liberate only a limited volume of air =

$$V\left(\frac{1}{r} - 1\right)$$ depending on the drop of pressure. In the foregoing example

the receiver volume was 109,200 cu. ft., with a 12-lb. drop and 90 lb. initial gauge pressure the volume of air liberated is

$$= V \times \left(\frac{1}{0.886} - 1\right) = V \times (1.113 - 1) = V \times 0.113$$

or only a little over 11 per cent. of the receiver volume.

In this respect the action of such air receivers (expansion tanks) is identical with that of a flywheel in an Ilgner transformer. In both cases only part of the stored energy can be liberated. There is, however, no loss incurred when storing energy in the form of compressed air if the receiver is tight, while in case of a fast running flywheel considerable power has to be consumed to overcome friction and windage, which power is absolutely lost. The action of the flywheel in this respect is similar to that of a leaky air receiver from which there is a constant leakage of such magnitude that, if filled with air, the air would in a few minutes be lost. An illustrative example of this will be given further on.

The foregoing shows how the instantaneous peaks, due to acceleration of hoists, were taken care of, but in addition to these instantaneous peaks there were longer peaks due to rapid periods of hoisting as illustrated by Fig. 2.

In order to get an idea of what peaks would arise from such conditions the writer estimated that if all the mines of the Butte district were connected to the power system and operated with compressed air, and if the operation could be so regulated that the periods of high air consumption would not overlap each other at the different mines, the resultant air consumption would be as per diagram Fig 16. This diagram is based upon investigation at different mines in the year 1909 and includes 27 hoisting engines of various sizes and capacities. According to this chart the highest rate of air consumption would be 37,200 cu. ft. free air per minute, and the average, 27,660 cu. ft.[1]

[1] It can be seen from this chart that there are 10 peaks in the air consumption curve, above the average and 10 depressions. The different peaks and depressions are in reality the average of a great number of peaks and depressions of very great magnitude, resulting from the simultaneous acceleration of several hoists and simultaneous periods of short duration during which the air consumption is very low. Some of these peaks figure as high as 100,000 cu. ft. of free air per minute during a few seconds of time. Such air consumptions, as have been stated before, are taken care of by the expansion tanks.

ESTIMATED AIR CONSUMPTION CHART
FOR 27 HOISTS
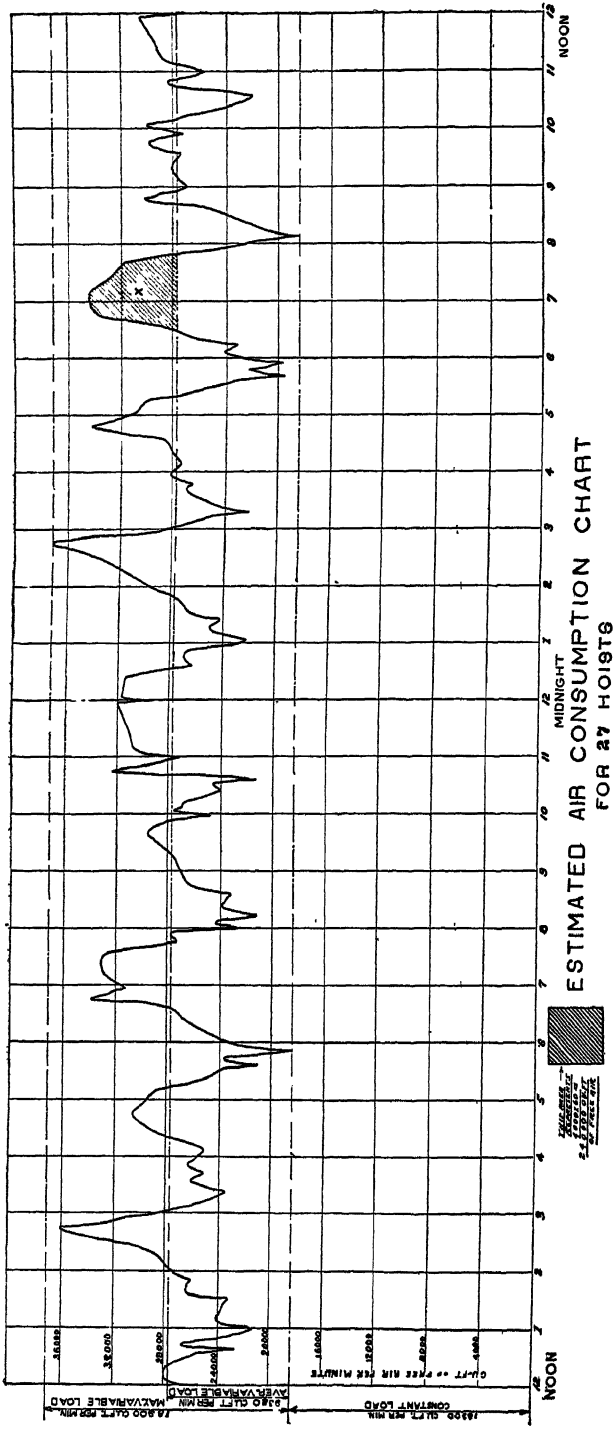
FIG. 16.—ESTIMATED AIR CONSUMPTION CHARTS.
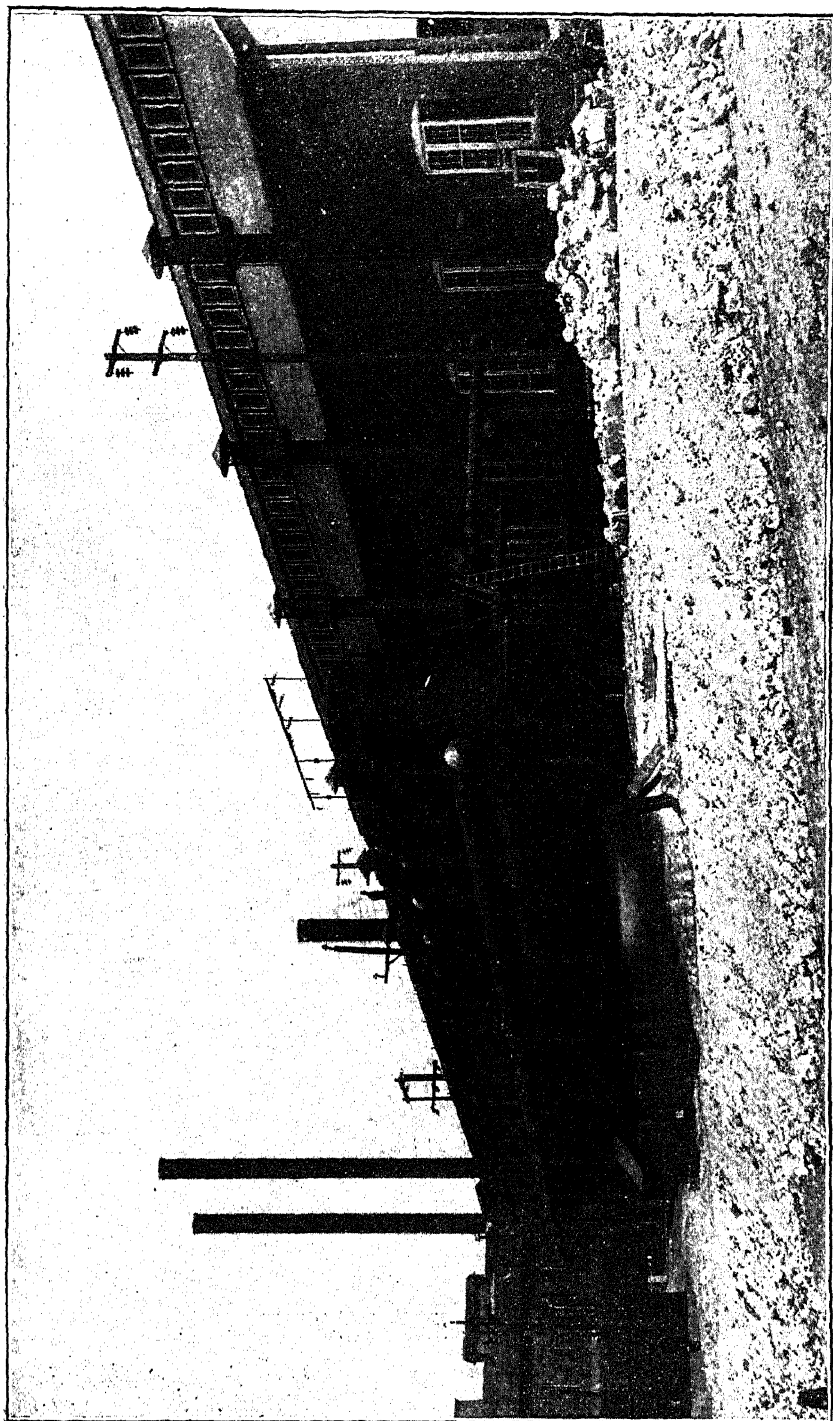
FIG. 17.—OUTSIDE OF COMPRESSOR PLANT.

This chart can be considered to represent the most favorable distri-
bution of the work with minimum peaks in the air consumption. The
largest peak (X) in the diagram represents a volume of 44,000 cu. ft. of air
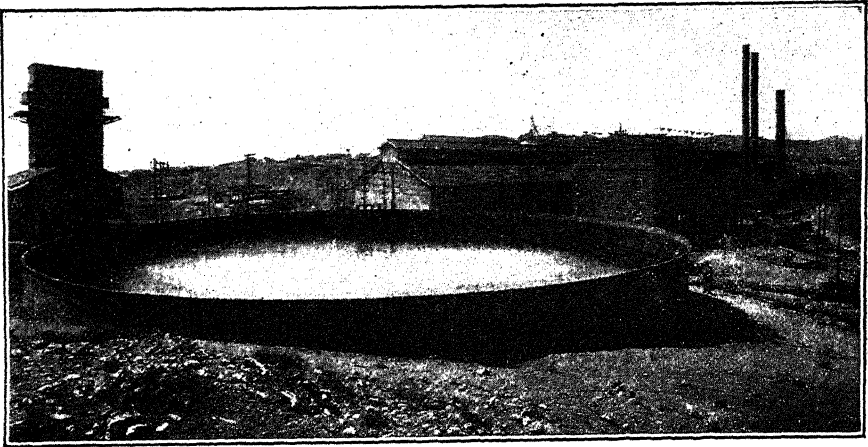
FIG. 18.—HYDROSTATIC TANK.
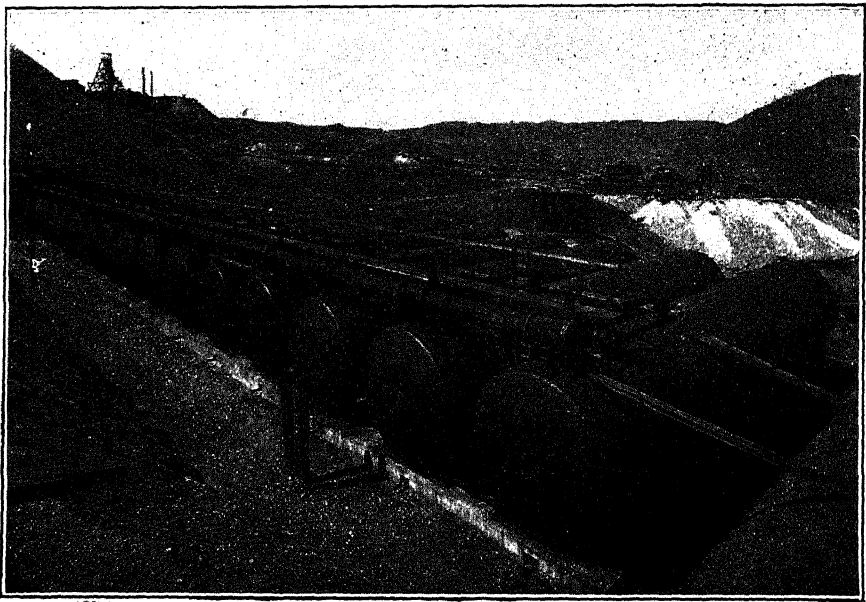
FIG. 19.—HYDROSTATIC AIR RECEIVERS.

at 90 lb. pressure and gives the measure of the minimum storage capacity
required in order to equalize the load on the system. The energy repre-
sented by this peak is about 1,060,000,000 foot pounds.
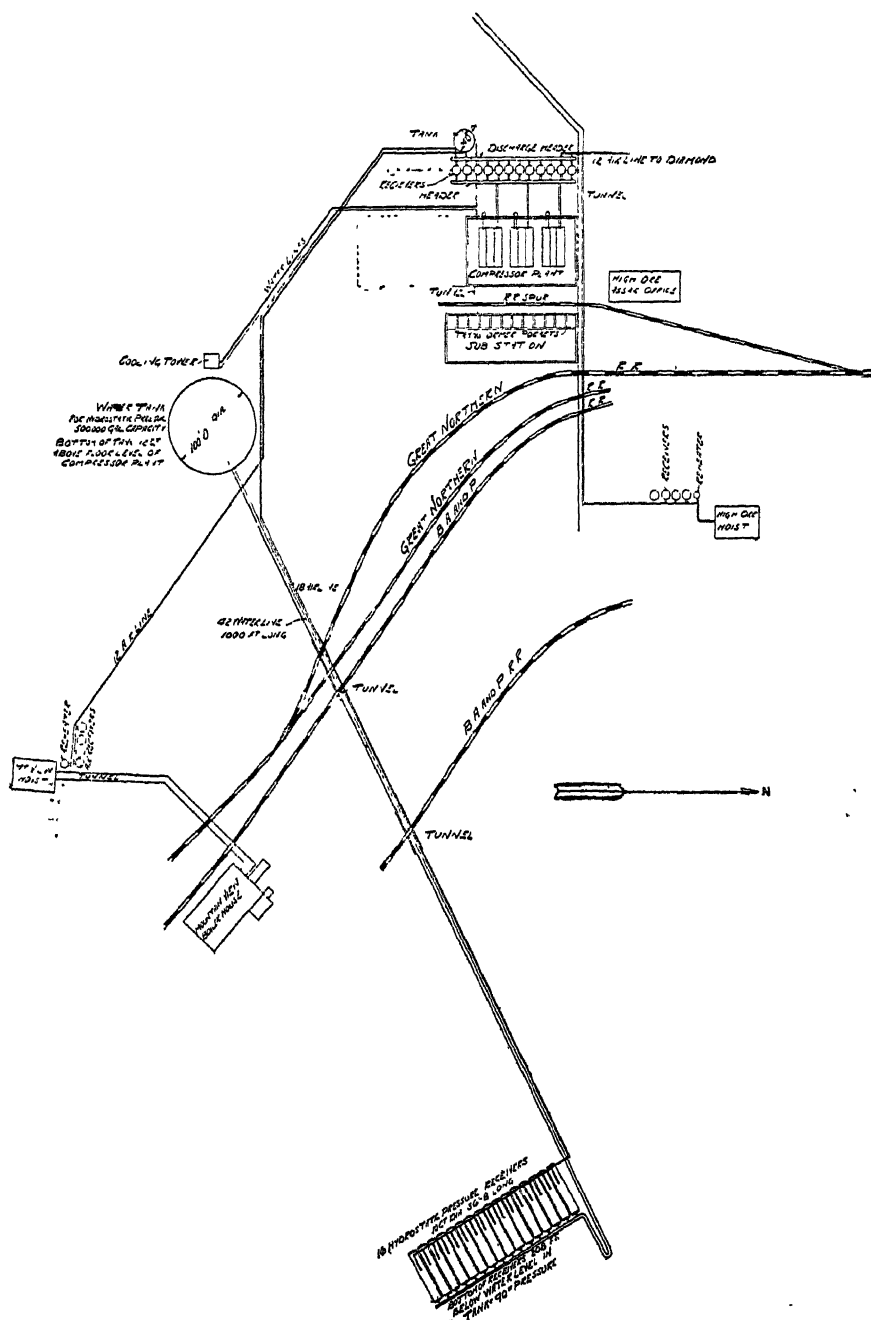
FIG. 20.—BUTTE HOIST COMPRESSOR PLANT AND HYDROSTATIC PRESSURE SYSTEM, BUTTE, MONT.

To provide for this peak of 44,000 cu. ft. capacity with expansion tanks would mean that with a pressure drop of 10 per cent. ($v = 0.9$) we would then have $(v_1 - v) = 44,000 = V(\frac{1}{r} - 1) = 0.1V$ or $V = 440,000$. Such a storage receiver would be prohibitively large.

If, therefore, a uniform load on the electric motors operating the hoisting engines is aimed at, most of the energy used must necessarily be stored energy; hence, the storage device must not only have great capacity, but must also retain the stored energy without loss. A body of water located at a certain elevation above a suitable motor would be an ideal device for storing power. The whole potential energy of it would be available for generating power, and as long as the reservoir and its connections are tight, the energy can be kept in storage for an indefinite period of time.

The hydrostatic storage plant forming part of the Butte power system consists of a number of air receivers connected so the whole forms a receiver of about 66,000 cu. ft. capacity. Some 210 feet above these receivers there is an open water tank 100 ft. in diameter. A pipe leads from the bottom of this tank to the bottom of the air receivers. Air from the compressor plant is piped to the top of these air receivers. Fig. 20 is a general drawing of this plant. Water is run into the elevated tank until the lower tanks and the rising main are filled up to the bottom of the elevated tank. When air is compressed into the lower tanks it drives the water out, and when 66,000 cu. ft. of compressed air have been pumped into these tanks the water is out and nearly fills the upper tank. A long return bend is formed in the water pipe, which bend runs down hill for a considerable distance, so that, if the water is driven out of the air receivers, the air pressure will have to rise very materially before it can force the water out around the bend and before air could enter the water pipe and blow out into the upper tank.

Fig. 18 shows the open tank on top of hill with the substation and compressor house in the background, and Fig. 19 shows the hydrostatic receivers at foot of the hill.

The diagram in Fig. 21 illustrates the capacity of this storage system. When the air receivers are full of air and all the water is in the upper tank we have stored up 66,000 cu. ft. of water under a head of 210 ft. The potential energy of this water is 865,141,200 foot pounds and is represented in the diagram by the shaded area A. In addition to this there is available for power the expansion force of air at 90 lb. pressure represented by the shaded area B in the diagram, the potential energy of which is 730,179,000 foot pounds. We have thus in this plant a total capacity of storage of 1,595,320,200 foot pounds.

It may be of interest to compare the capacity of this storage plant with other devices used for the same purpose. Thus we have shown that air

receivers placed on the compressed air pipe line system and liberating energy by drop of pressure, will with 12 per cent. drop in pressure liberate about 11 per cent. of the volume of air contained in such receivers.
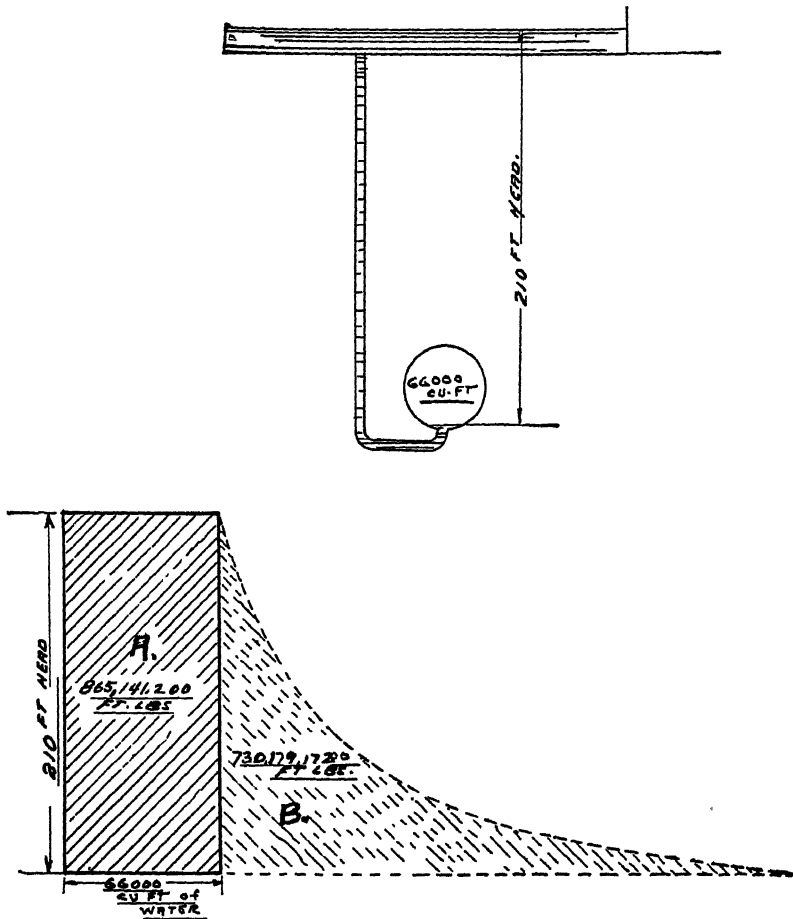
FIG. 21.—DIAGRAM SHOWING HYDROSTATIC SYSTEM.

In order, therefore, to liberate 66,000 cu. ft. of air, the receivers should have a volume of

$$\frac{66000}{0.11} \text{ or } 600,000 \text{ cu. ft.}$$

The flywheel transformers used at the Matthias Stinnes Mines at Essen, in Germany, and which are among the largest built so far, have two flywheeels weighing 44 tons each and running at a rim velocity of about 17,000 ft. per minute maximum. In delivering power the speed is allowed to drop about 18 per cent. The maximum speed is about 284 ft.

per second. The radius of gyration of these wheels is about 0.8 of the radius of outside of rim, so that the masses of the wheels act with a maximum velocity of 0.8×284 = 227 ft. The energy of the revolving wheel at full speed is thus:

$$\frac{2 \times 88000}{2 \times 32.17} \times 227^2 = 140,000,000 \text{ ft. lb. approximately.}$$

A drop of speed of 18 per cent. brings the minimum velocity at radius of gyration down to 0.82 × 227 = about 186 ft. per second. The potential energy in the wheel at that speed is thus:

$$\frac{2 \times 88000}{2 \times 32.17} \times 186^2 = 94,000,000 \text{ ft. lb. approximately.}$$

The available energy stored is thus 140,000,000 − 94,000,000 = 46,000,000 foot pounds. The statement was made to the writer at the mines that it required over 300 h. p. to keep up the full speed on this apparatus when running idle, 240 h. p. would then be a safe estimate of the average power if the speed drops 18 per cent.

240 h. p. is 240×33,000 = 7,920,000 foot pounds per min., dissipated energy, so that the whole amount of stored energy would be lost in

$$\frac{46000000}{7920000} = 5.8 \text{ minutes.}$$

George McCulloch, in his book on winding engines, states that the Ilgner transformer at the Deutscher Kaiser Mine requires 95 kw. to run idle. This transformer is said to have a 46-ton wheel running 370 rev. per min. The rim speed is not given but it is stated that the maximum kinetic energy of the wheel is 81,600,000 foot pounds. The power required and the energy of the wheel are nearly in proportion to that given above for the larger transformer at the Matthias Stinnes Mines. In that case it was found that about one-third of the total maximum energy could be liberated. Let us assume that in case of the transformer wheel at the Deutscher Kaiser we also can liberate one-third of the maximum energy or about 28,000,000 foot pounds of work. The power to keep up the full speed was in this case 95 kw. or an average of 77 kw. if this speed drops 18 per cent.

$$= 77 \times 44,240 = 3,400,000 \text{ foot pounds per min.}$$

The stored energy is thus dissipated in $\dfrac{28000000}{3400000}$ = about 8.25 min.

A compressed air storage of any type is not subject to leaks of such magnitude and is therefore better adapted to the conditions of hoisting as they exist in most ore mines where the work is irregular and the load factor very low. The storage system used in Butte is no doubt the best as the whole energy stored can be drawn upon. At Butte the air receivers

forming part of this storage had to be made of steel. In many localities the air receiver can be built into the country rock and in that case even a much larger volume of air can be stored than in the plant described above. Mr. MacNaughton is now constructing an underground air storage receiver at the Calumet & Hecla Mine which will have a volume of 252,000 cu. ft. It will be nearly four times as large as the Butte air storage plant. With a 10 per cent. drop of pressure ($r = 0.9$), such receiver will liberate $252,000 \times 0.1 = 25,200$ cu. ft. of compressed air. It will be the largest air storage receiver thus far built in the world.

A test was made two years ago to determine the efficiency of the Butte air storage system. The storage system was filled with air and the compressor stopped. The Mountain View hoist was then run for an hour *with stored energy,* hoisting regular loads of 5 tons in balance. 21 skips were hoisted from a depth of 1,870 ft. and 16 skips from 1,570 ft. depths. The drop of the water level in the 100 ft. tank was then measured. Then the compressor was started and the air drawn from the system during the run, which was of one hour's duration, was restored.

The electrical input to compressor for restoring the air was ascertained. Indicator cards were taken during the run, which showed a very low mechanical efficiency of the combined engine and hoisting gear, due to the fact that the Mountain View shaft was crooked, causing binding of the skips. The mechanical efficiency was here only 69 per cent. when hoisting from 1,870 ft. and 57 per cent. hoisting from 1,570 ft. depths.

The result of the test was that 327 h. p. at this shaft required an electrical input to the motor of compressor of 836 kw. or 2.56 kw. per shaft h. p. Correcting this for the excess friction of the Mountain View shaft over that found by test at the High Ore and Speculator shafts, in both of which the mechanical efficiency of the hoisting engines and skips was about 80 per cent., we get an input of 2.04 kw. per shaft h. p. = 36.5 per cent.

This test was made with very moderate heating of the air (air temperature 258°) and the retardation at the end of each run was done by "plugging" the engine in the old way and not by compressing air back to the system. Had air temperature been 370° the efficiency would have been 42.2 per cent.

The Air Heaters.

The air reheaters on this power system are of an indirect type, in which steam of 200 lb. pressure is used as the heating medium. In nearly all of the mines there is a steam heating plant for the dry house. The small boiler that furnishes steam for the air heater is placed in the boiler house with the boiler for the dry house and one fireman can take care of both. In most cases the air heater is located at a higher elevation than the boiler house so that the water of condensation from the air heater runs by gravity back to the boiler as soon as it is formed.
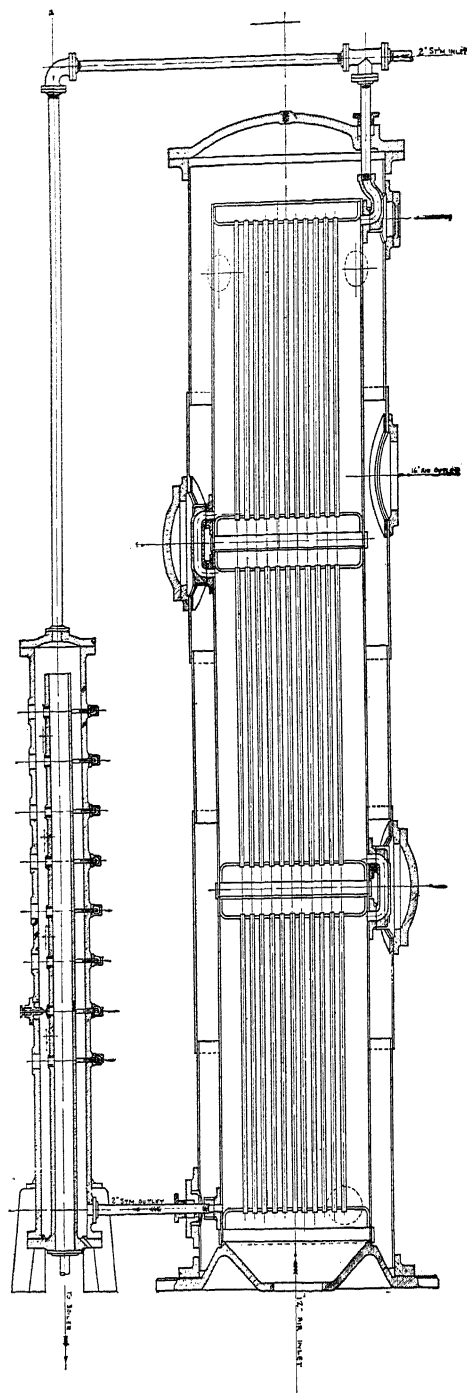
FIG. 22.—DETAIL OF AIR HEATER.

Fig. 22 shows a section of the air reheater, and Fig. 23 the manner in which it is connected to the boiler. The heater consists of an outer vertical cylinder in which are placed three cylindrical shells closed at the ends by tube heads. A number of tubes are expanded into these tube heads, which tubes run through the shells. The three (3) inner shells are mounted one above the other, the lowest one resting on a ledge formed in the outer cylinder. The air enters at the bottom, passes through the tubes of the inner shells and sweeps over the outer surface of the upper
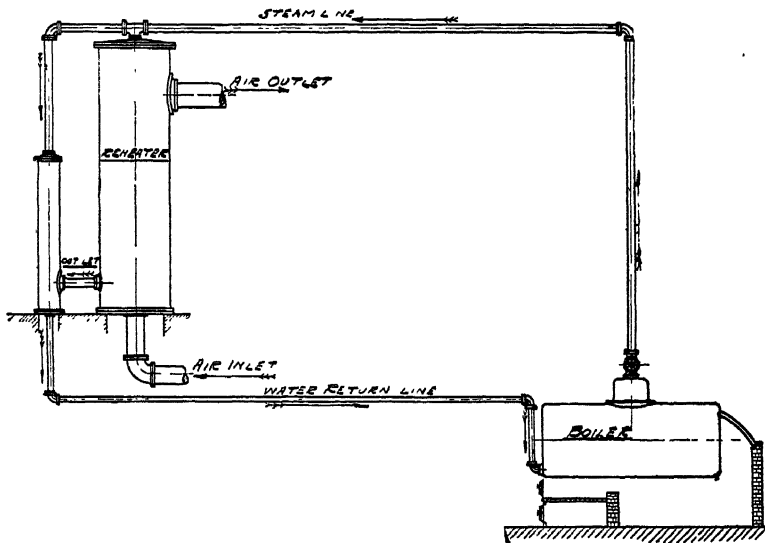
FIG. 23.—SYSTEM FOR REHEATING AIR.

one of these inner shells before it passes out of the heater. The inner shell is filled with steam of 200 lb. pressure which surrounds the tubes. The steam enters the upper one of the inner shells near its top. Balancing pipes are fitted between the bottom of the upper and top of the middle shell, and between bottom of this and top of lower shell. The water formed by condensation leaves the lower shell at the bottom. It has been found that the water of condensation is a very efficient medium for transmitting heat to air, in fact, more efficient than the steam. It is, therefore, important to regulate the water level so that the temperature of the outgoing water is a minimum. For this purpose the heaters are provided with a water column inside of which is a vertical pipe connected to the water discharge on the lower inner shell of the heater. There are several openings at different heights in the vertical pipe, perforating its walls and provided with stop valves so that any of these openings can be closed. At the lowest point of the outer pipe, or water column, is the discharge nozzle which is piped to the boiler. By closing the dis-

charge holes in the inner pipe in succession, beginning with the lowest one, the water level in the inner heater shells can be elevated to the desired point.

The Hoisting Engines.

The largest producing mines in the Butte district are the following:

Mountain View	West Stuart
High Ore	Pennsylvania
Diamond	Tramway
Original	Leonard

Of these the Mountain View, High Ore and Diamond use flat ropes and reels; the others use 12 ft. drums with 1½-in. round ropes. The power requirements of the different mines of this group are very nearly identical. As these engines were in good condition, and of ample strength in all parts, the only change necessary for adapting them to the economical use of compressed air was in the cylinder construction and the operating gear.

At each of these mines and also in other mines operated by the Anaconda Copper Mining Co. there were auxiliary hoists, several of which did more lowering than hoisting. These smaller hoists were discarded and replaced with new ones so designed that the downgoing load compresses air and returns it to the storage plant. 12 new hoists with cylinders 28 in. by 48 in. and 6 ft. drums were built, four of which were provided with double clutched drums. All these hoists have a winding capacity of 3,500 ft.

Figs. 24 and 25 show one of the new air cylinders fitted to the main hoists. Nine sets of such cylinders were made, all being 34 in. diam. by 72 in. stroke. Fig. 26 gives a view of the single drum auxiliary hoists, and Fig. 27 gives a view of the double drum auxiliary hoist. These views were taken at the works of the Nordberg Mfg. Co., where the machinery was built.

I think that this machinery is the first attempt to build large air motors designed to use compressed air, so as to approach the efficiencies that theoretically should result when air is compressed, heated and expanded. Theoretically one cubic foot of free air at the atmospheric pressure at Butte requires 4,356 foot pounds of energy for compression to 102 lb. absolute, or 90 lb. gauge pressure if the work was done in a perfect 2-stage compressor. The total mean effective pressure of this process is 30,248 lb. per sq. in. The temperature of air discharged from this compressor is 250° F. if the initial temperature is 60° F.

A perfect two-stage compressor is one without clearance and without pressure losses. The air is cooled to the initial temperature (say 60° F.)

before entering high-pressure cylinder. The cylinders are so proportioned that one-half of the work is done in each.

If the heat of compression is lost so that the temperature of the compressed air drops back to the initial temperature of 60° and the air be expanded without taking up heat in a cylinder to atmospheric pressure, a work of 2768.5 foot pounds is performed. The efficiency of the process is, therefore, 63.5 per cent.

By heating the air before expansion and thereby increasing its volume, the air can be made to perform more work and the efficiency will be in-
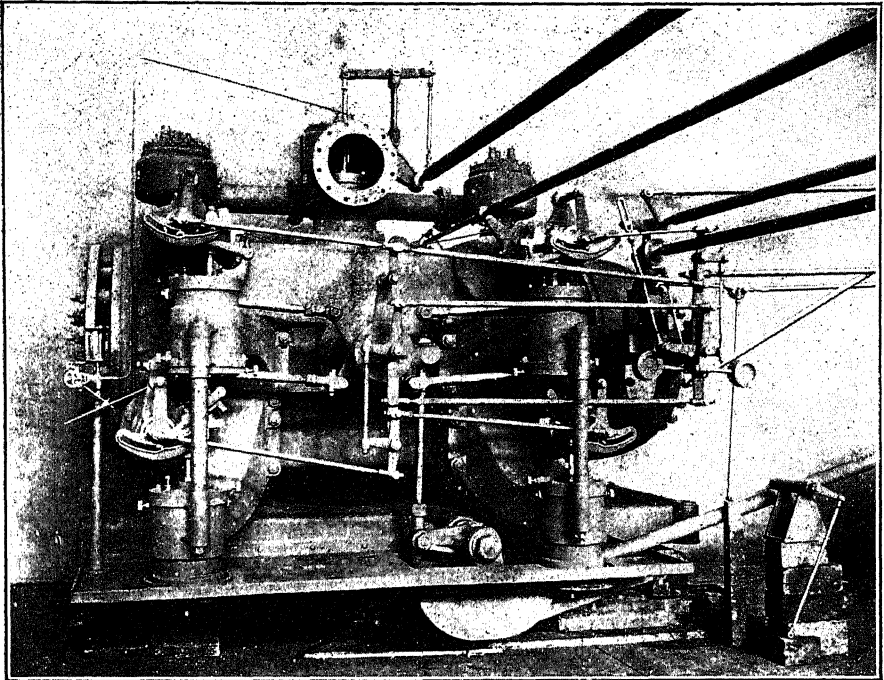
FIG. 24.—CYLINDER FOR AIR HOIST.

creased. If the air was heated to 355° F. it would perform 4,356 foot pounds of work or just as much as was required for its compression. The efficiency would then be 100 per cent.

To obtain 100 per cent. efficiency under above conditions would cost 0.053 c. per h. p. per hour if the fuel (coal) was worth $4 per ton, containing 12,500 B.t.u. per pound and the efficiency of the air heater was 60 per cent.

If the assumed quantity of air (1 cu. ft. at atmospheric pressure and 60° F.) after compression was expanded in a two-stage motor with cylinders displacing the same volume of air per unit of time, as the compressor,

the work done in such a motor would be exactly the same as that required to compress the air, if the air entered both cylinders of the motor with a temperature of 250° F. The motor would therefore require an interheater between high and low pressure cylinders and a preheater if the heat of compression was lost. By such a process 100 per cent. efficiency is obtained with only 250° air temperature and at an expense of 0.068 c. per h. p. per hour.

In the above it is assumed that there is no loss of pressure between the compressor and the motor. Should there be a drop of 15 lb. between the
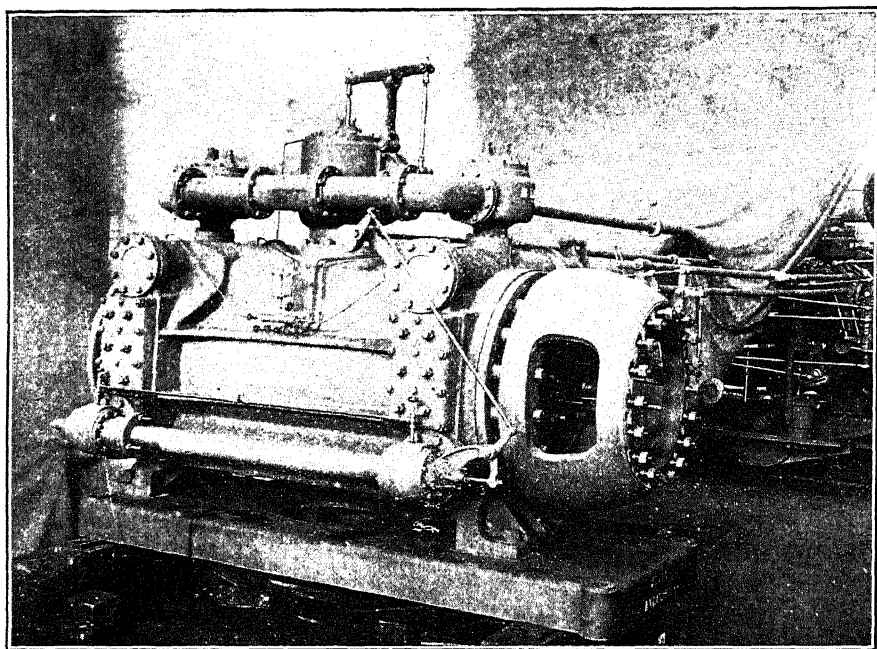
FIG. 25.—CYLINDER FOR AIR HOIST.

discharge pressure of the compressor and the initial pressure on the motor, we would, in case of a two-stage motor, have to heat the air before entering the high pressure cylinder to a temperature of 263.4° F. and to 289° F. before entering the low-pressure cylinder. The air will then be exhausted at a temperature of 94.9° from the low-pressure cylinder, which is 34.9° higher than the initial temperature of compression. The cost of reheating in this case will be 0.077 c. per h. p. per hour if the efficiencies and fuel cost and quality are as assumed above.

The air pressure in a power system at a mine is for practical reasons determined by that used on the rock drills. At Butte this pressure is 90 lb. From the above it follows that by reheating the air, the largest
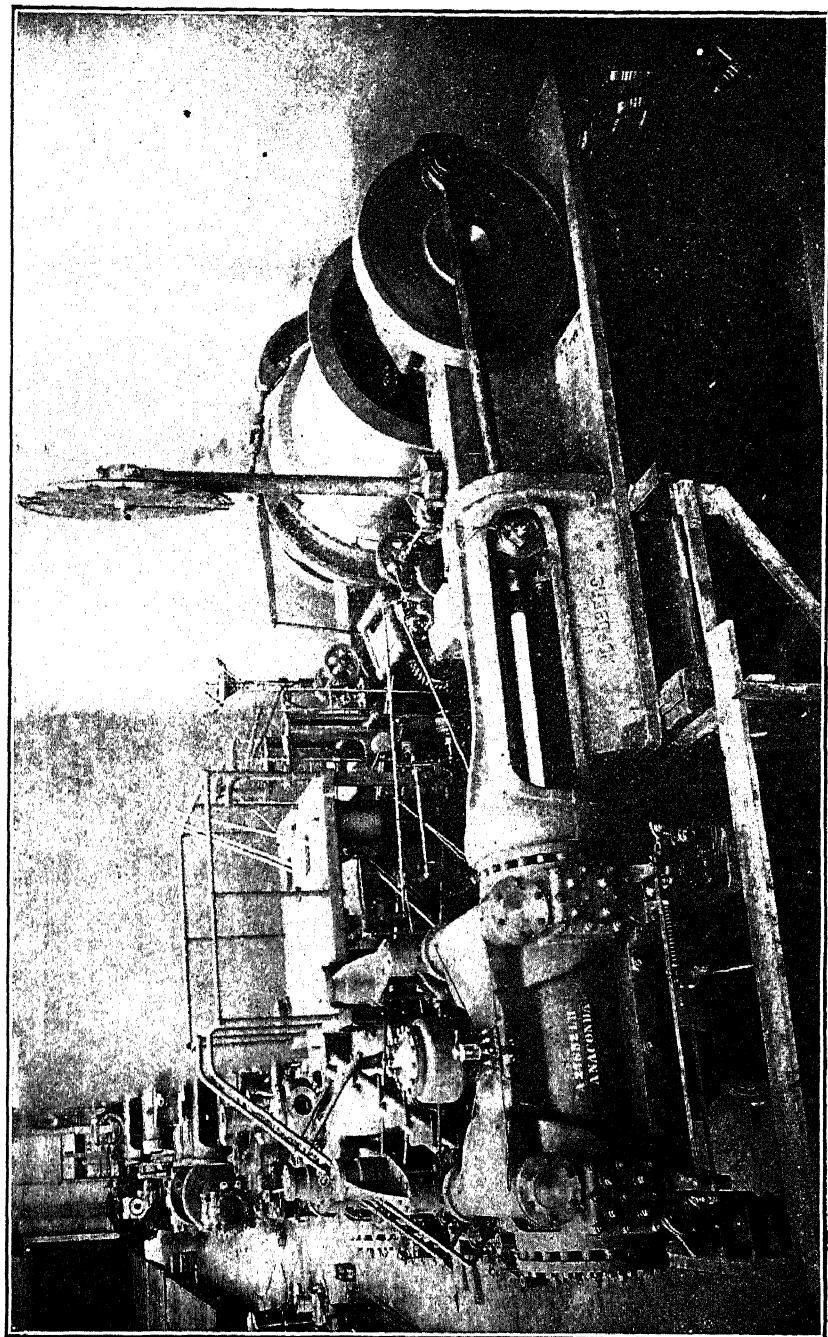
FIG. 26.—GENERAL VIEW OF AIR HOIST.
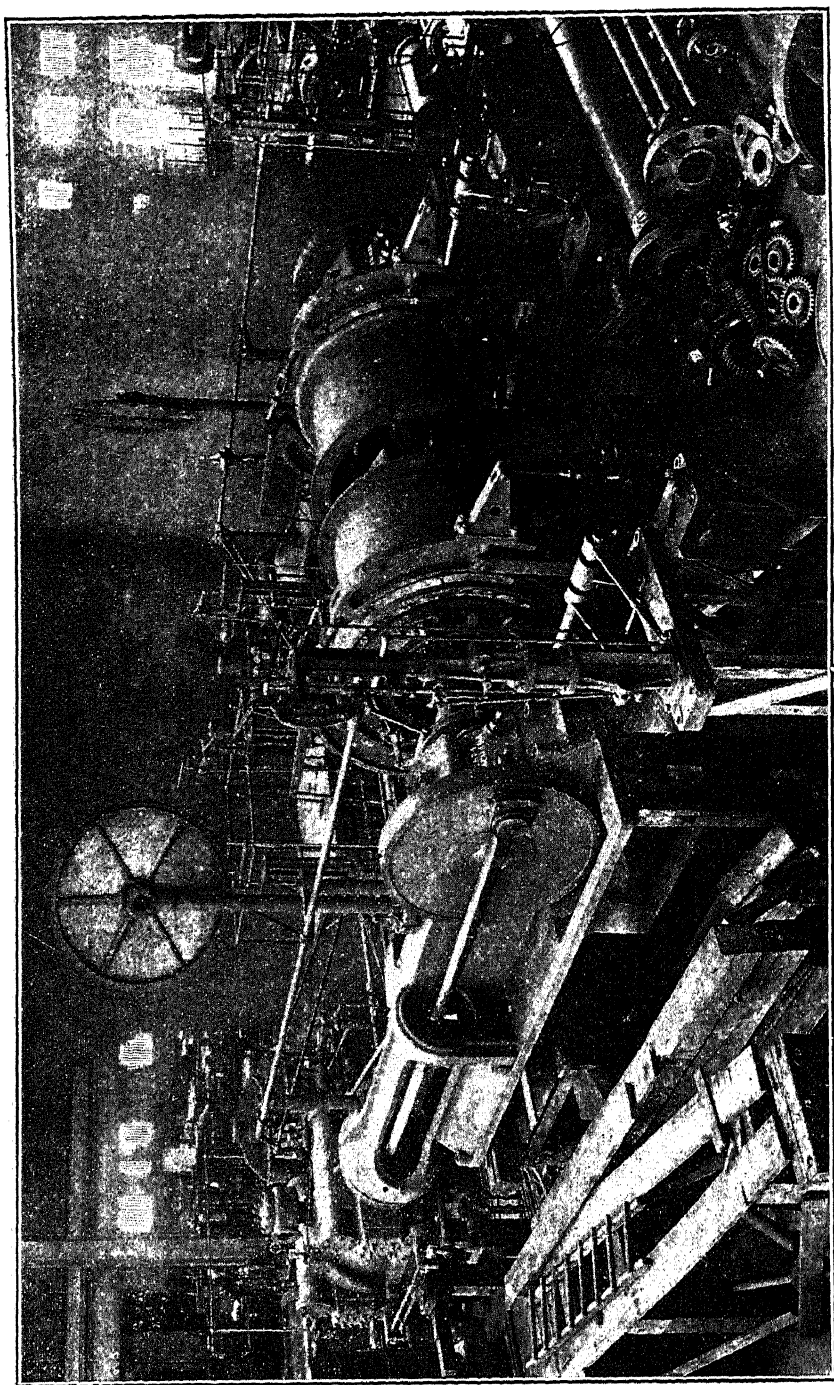
FIG. 27.—GENERAL VIEW OF AIR HOIST.

possible gain in efficiency is 36.5 per cent. and it will be noticed that the expense in fuel for gaining this efficiency is quite a small item.

In order to obtain maximum efficiency on the air-operated engines there must be no drop of pressure between the compressor and the engine, the engine must have no clearance, the air must always be expanded to the atmospheric pressure and the cut-off should be abrupt so there is no wire drawing.

The pressure drop can, with proper pipe lines, and receivers such as are used in the Butte plant, be kept inside of 10 per cent. of the initial pressure.

The clearance in an air motor, the piston speed of which runs up to nearly 1,000 ft. per minute, as is the case with the long stroke hoists at Butte, will of necessity be large, as such engines require large valve ports, much larger than a steam engine at the same speed, for the reason that the compressed air has a greater density than steam of the same pressure. If, however, the exhaust valves are made to close early enough during the return stroke of the piston to compress air to a pressure equal to that at the admission, then the effect on the efficiency of the clearance space, no matter how large it may be, is entirely eliminated. In a hoisting engine the compression would make it impossible to maneuver at slow speed. To overcome this the Butte hoists have a valve gear so designed as to automatically adjust the closure of the exhaust valves to compress to initial pressure as soon as the engine has attained a high enough speed to safely carry such compression, and to automatically remove the compression when the engine drops below this speed. We have, therefore, in the Butte hoists no clearance loss except when running at slow speed. The clearance loss, if present, would result in a reduction of efficiency of about 25 per cent.

If the load on an air motor and the air pressure was absolutely constant, then the cylinder dimensions could be so proportioned that the cut-off takes place at the exact point expanding from which the terminal pressure coincides with that of the atmosphere. In a cylinder without clearance this point of cut-off would be about 22 per cent. of the stroke if the absolute pressure of the atmosphere is 12 lb. and the air pressure 90 lb. above the atmosphere. In a cylinder with clearance the cut-off takes place earlier in the stroke to fulfill this condition. At any other point of cut-off there is a loss of efficiency. A later cut-off than 22 per cent. produces incomplete expansion. In this case the air is exhausted from the cylinder at a pressure higher than that of the atmosphere, causing a waste of energy which, if the cylinder were allowed to take air at full stroke, reaches a maximum of about 50 per cent. of the total energy in the compressed air. If the cut-off is earlier than 22 per cent., the air will expand below the atmosphere, producing "loops" in the indicator cards. Such "loops"

are areas of negative work and as air expands adiabatically, the pressure during expansion dropping at a much greater rate than the volume swept by the piston increases, these areas of negative work become very much larger than in case of a steam engine.

In the air hoists at Butte the formation of such loops in the indicator cards is prevented by admitting atmospheric air into the cylinder as soon as the pressure of the expanded air reaches the atmospheric pressure. At all points of cut-off that are not so late as to produce incomplete expansion, the air is thus used at best efficiency.

The loss due to incomplete expansion is one that in a hoisting engine can be minimized only by increasing the volumes of the cylinders. It cannot be entirely eliminated.

In starting a hoisting engine from a state of rest, it is of great importance that the admission valves close as late as practicable in order that the engine may start readily from any position. If these valves close at 0.9 of the stroke, there is one position of the cranks in the ordinary quarter crank engine where only one of the two cylinders can take steam, and in that position the angle between the crank center and that of the connecting rod is such that only about 0.65 of the crank radius is active. The available starting torque is thus the pressure on one piston multiplied by 0.65 of the crank arm. The active crank arm decreases rapidly with shorter cut-off.

This condition is much improved in the 4-cylinder design of hoist such as built by the Nordberg Mfg. Co. for Tamarack, Calumet & Arizona and Tuolumne mines. In that type there are in the most disadvantageous crank position three cylinders out of four available for starting and a shorter initial cut-off can be used without affecting the ease of maneuvering.

The acceleration of the hoist also requires late admission; in fact, in any type of hoist air at nearly full stroke will have to be used during several revolutions of the engine. The hoisting of men is preferably done with the engine throttled and without cut-off. The loss in efficiency due to incomplete expansion is the most serious loss in an air-operated hoist. If there was a way to eliminate it, the work performed per unit of weight of air used by the engine would be very nearly the same as required to compress the same quantity of air if the heat of compression is restored by reheating, which, as has been stated above, can be done at a very small expense.

By the use of a two-stage motor the losses from incomplete expansion can be very much reduced. At a maximum filling of the cylinders of 0.9 of the stroke the loss from incomplete expansion is about one-half of what it would be in a single-stage motor or about 25 per cent. of the energy in the air entering high-pressure cylinder if the initial air temperatures in high and low pressure cylinders are equal.

Such a two-stage motor with two high and two low pressure cylinders should preferably be so designed that the time of the eight impulses produced by the cylinders is equally divided in the revolution, so that at every one-eighth of the revolution one of the pistons passes the dead center. In a properly proportioned two-stage air motor the impulse waves in the torque diagram are exactly the same for the low-pressure as for the high-pressure cylinders, so that the torque diagram has the same appearance as that produced if all cylinders were of the same size and the air expanded in a single stage. In an engine of this type, whether single or two-stage, it is not necessary to extend the cut-off to nine-tenths of the stroke, but the engine will start and maneuver easily with the admission valves set to close at a little less than 0.7 of the stroke. The loss from incomplete expansion in a two-stage motor of such design and proportions will be only 8 per cent. at 0.7 per cent. admission. The use of a two-stage motor thus gives a way to practically eliminate the loss in efficiency due to incomplete expansion. When the load conditions are such as exist in deep shafts of ore mines, such a motor will produce economical results with compressed air that could not be approached by any other mode of transmitting power.

It has already been stated that in several of the Butte mines the operators habitually retard the hoists by "plugging" instead of by the brake. "Plugging" is a process where the hoist is reversed at the time when retardation commences. The throttle is closed and the pistons of the hoist compress the air or steam entrapped back of them. The intensity of the compression is regulated by manipulating a bypass valve through which the air or steam is allowed to escape to the atmosphere. The work performed by the pistons during this process is absolutely lost and the only advantage of using it is that it saves the brakes. Frequently when plugging an engine, very high pressures are brought upon its running gear and frames.

In designing the new cylinders for these hoists, arrangements were made whereby the retardation is effected by compressing air back into the pipe line and receiver system. This is also done when, as frequently is required, loads are lowered into the mine. The compression can be regulated exactly as the expansion, i. e., the quantity of air compressed can be varied to suit the load. The manipulation of the compressor gear differs little from the manipulation of plugging to which the operators are accustomed.

In rebuilding these hoisting engines for operation with compressed air the principal parts of the old engines had to be retained. This was somewhat of a handicap as none of these engines was strong enough to admit of such cylinder diameters as we would like to have used. The High Ore and Diamond engines had cylinders 30 in. x 72 in. and the one

at Mountain View 28 x 72 in. The new cylinders were made 34 x 72 in. in all these cases. These sizes were about as large as the frames of the Diamond and High Ore hoists could stand and in case of the hoist at Mountain View, new and stronger frames were made.

The above-mentioned hoists are all fitted with reels for flat ropes and were the first ones converted. The drum hoists at Original, West Stuart, Pennsylvania, Tramway and Leonard had cylinders 32 x 72 in. when run with steam. These were provided with cylinders 34 x 72 in. All the large hoists thus have cylinders of that size. The main hoist at West Colusa originally had steam cylinders 20 x 60 in. The new air cylinders were made 23 x 60 in.

While, as has been intimated above, a somewhat higher efficiency could have been obtained with these hoists had it been practicable to use larger cylinders, still the efficiency is very satisfactory considering the here existing conditions of extremely varying loads. There has been no complete test made to determine what the exact efficiency is and it would hardly be possible to make such tests, which, in order to give exact results, would have to be made with the system as a whole. A test run with stored air on the least economical of the large hoists showed that 1.62 kw. was consumed per indicated horse power. By comparing the indicator cards of the more economical hoists with cards taken during this test we find that these hoists would consume from 1.5 to 1.4 kw. per indicated horse power.

A detailed description of the mechanism used in the Anaconda air hoist may be of interest.

The cylinders and valve gear on these hoists must perform the following functions:

In starting the hoist the throttle is opened wide and air is admitted during nine-tenths of the stroke and exhausted during the entire return stroke, producing an indicator card as shown by a, Fig. 28.

The hoist is now accelerated and after having made from one to three revolutions, the governor takes control, cutting off the air supply at different points of the stroke as determined by the rise of the governor. Indicator cards as per b are then produced at this stage of the operation.

When about six-tenths of the full speed is attained the point of closure of the exhaust valves is advanced to such a point as to cause the air contained in the clearance spaces to be compressed to the full initial pressure. As the speed increases the point of cut-off at the air inlet valves is also advanced by the governor and indicator cards as per c are produced.

When the governor has risen to such a height as to produce the last card in the series c, the air is expanded to atmospheric pressure and theoretically the engine works at best efficiency.

In order to maintain this efficiency when the governor rises still higher
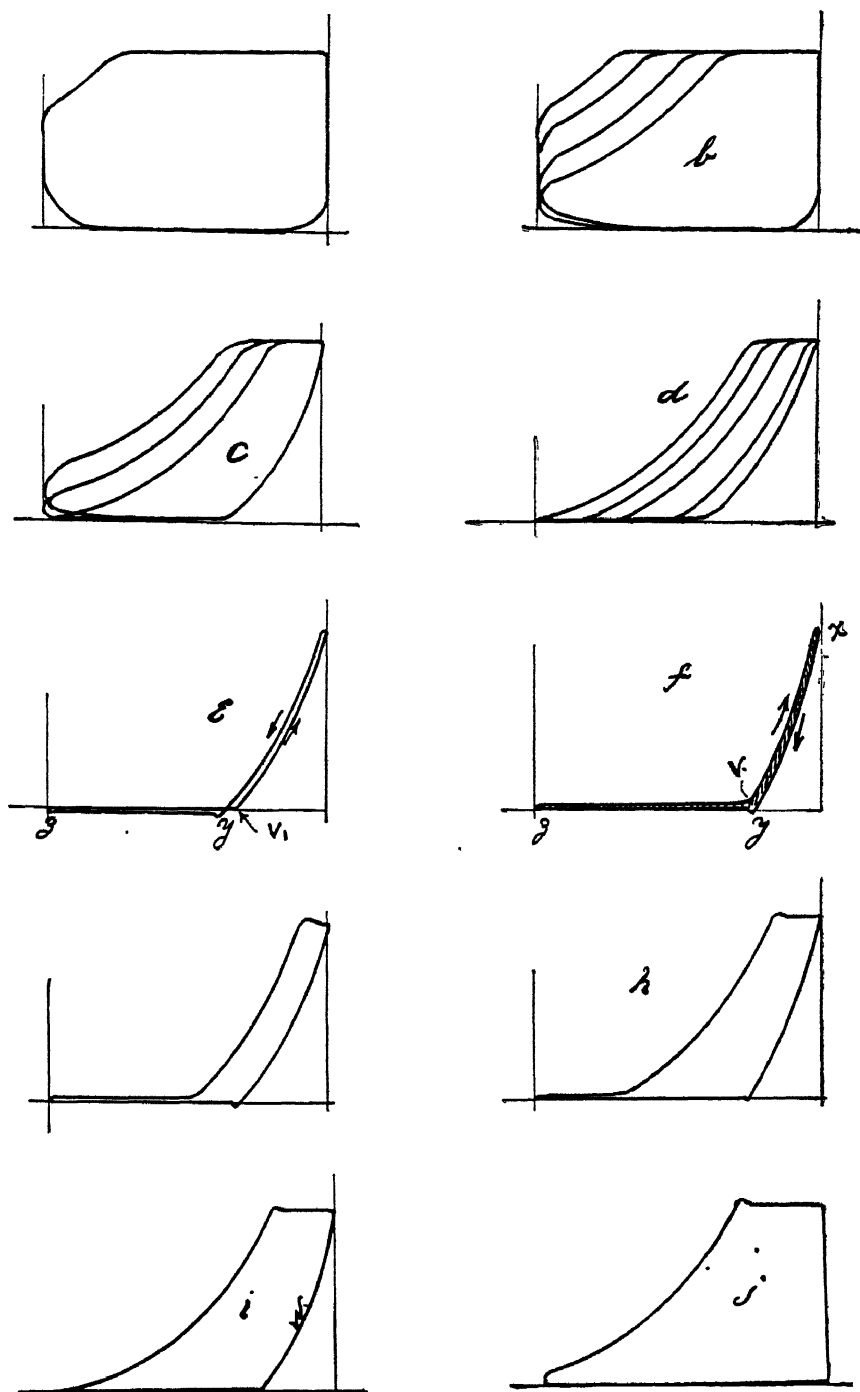
FIG. 28.—AIR HOIST CARDS.

as the load decreases, and in order to eliminate loops in the indicator cards under this condition, atmospheric air is admitted to the cylinder at all points of cut-off shorter than the last one in series c. This stage in the operation is shown in the diagram group d.

e shows the last diagram of the group. The air is here cut off practically on the dead center of the engine, expanded to atmospheric pressure, which pressure is maintained in the cylinder from the point of the indicator card at which the expansion line touches the atmospheric line up to the end of the stroke, and on the back stroke, to the point where recompression begins. This is the indicator card corresponding to a work = ± 0.

While the above-described functions are performed by the valve gear of the engine, the hoist has attained its maximum speed and is now running
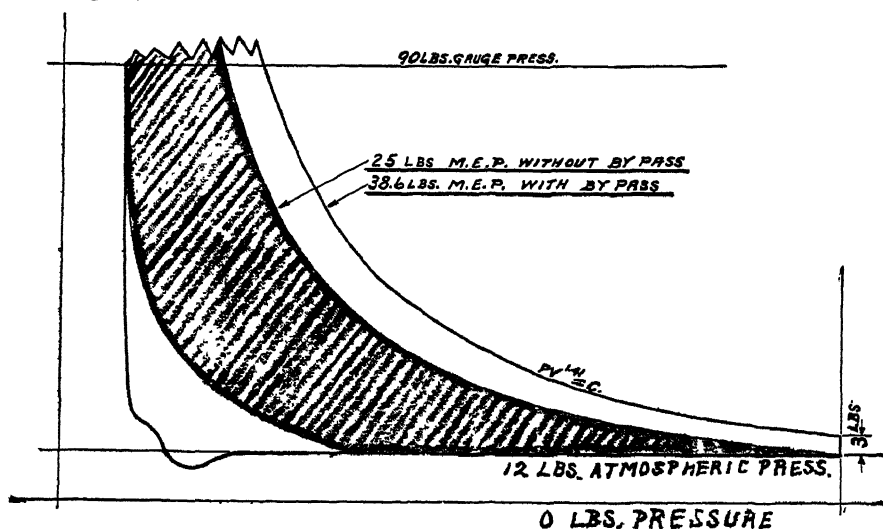
FIG. 28k.—COMPRESSION CARD FOR AIR HOIST.

under the influence of the momentum of the revolving parts and the masses attached to the ropes.

Under certain conditions of speed, friction and balance, the air could now be shut off and the hoist allowed to "run out," the masses being retarded by the friction and the increased static moment of the ascending load (if it is a reel hoist). These conditions are not frequently found so that generally the following additional functions have to be performed by the valve gear:

The hoist is to be retarded without applying the brake. The operator moves the regulating lever (which may be combined with the throttle lever) into retarding position. Thereby he causes the air-admission valve to remain closed throughout the forward and return stroke as they were when card e was produced. The only air present in the cylinder is that in
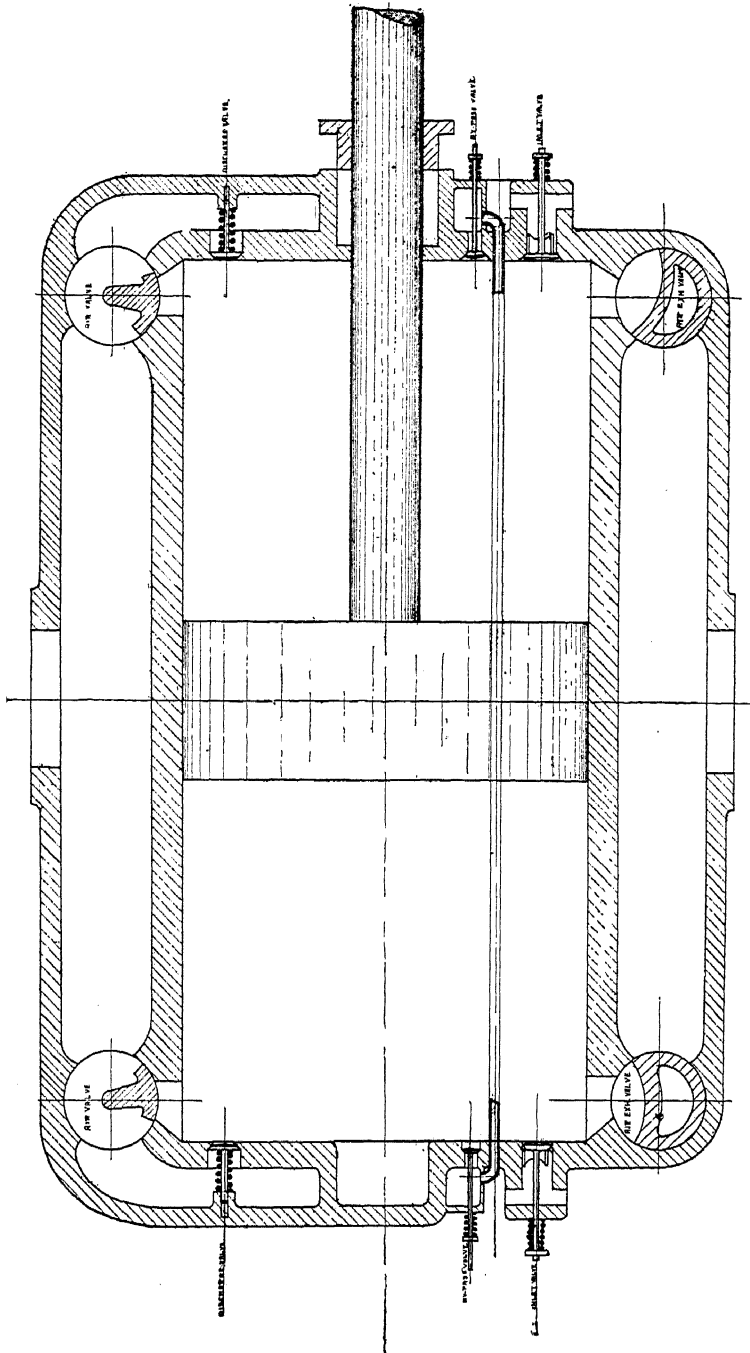
FIG. 29.—DETAIL OF AIR HOIST CYLINDER.

the clearance space which now expands as the piston progresses on its forward stroke from x to y, card f, and a further progress of the piston is accompanied with admission of atmospheric air into the cylinder from y to end of forward stroke z. The exhaust valve now opens and air is ejected during the return stroke. So far the process in producing cards e and f is identical. The regulating lever referred to above actuates the closing mechanism of the exhaust valves in such a manner as to vary the point of closure of the valves during the return or exhaust stroke of the engine. Thus, in the card f, produced when the regulating lever is barely moved from its neutral position, the point of closure of the exhaust valve has been advanced to point v in card f from point v_1 in card e. The result is the shaded area of the diagram which represents a certain negative work. By further movement of the regulating lever away from its neutral position the compressor cards g, h, i are made in succession, i representing the compression of the full volume of the cylinder. The effect of this mechanism is to exhaust from the cylinder all air that is not to be compressed and then to compress the balance. The valve gear has such a range that any volume from the full piston displacement to zero can be compressed, and as said before i represents the indicator card produced in compressing the full contents of the cylinder. By moving the regulating lever into its extreme position, an auxiliary valve is actuated whereby communication is established between both ends of the cylinders at the time when the piston passes the dead center and while both exhaust valves are closed. This causes the compressed air filling the clearance space behind the piston to flow over to the other side, which during the previous suction stroke has been filled with air at atmospheric pressure. The result is an increase of the pressure in front of the piston of several pounds and the disappearance of the reëxpansion line w (card i) and a much higher mean effective pressure of compression which brings the hoist to a standstill.

j is the indicator card produced at this the final stage of the operation. Fig. 28k shows the increase of mean effective pressure by applying the just described device. The shaded area in this diagram is that produced when compressing the full displacement of the air cylinders and reëxpanding the air in the clearance space in the regular way. It will be seen that by this device the mean effective pressure is increased from 25 to 38.6 lb. to the square inch.

The mechanism for performing the functions above described are illustrated in Figs. 30, 31, and 32. The cylinders are fitted with Corliss valves for admission and exhaust, but in addition to these (the motor valves) there are air inlet and discharge valves of self-acting poppet type which come into action when the machine compresses air back into the air system.

Fig. 29 is a diagrammatic section of these cylinders and shows the interrelation of the different parts of said cylinders. The usual bypass valves

for establishing communication between the two ends of the cylinder are also provided on these cylinders and are indicated in the diagram. As has already been mentioned, special use is made of these valves to intensify the compression in stopping the hoist. Corliss releasing valve gear is used on all the engine valves on the admission as well as on the exhaust valves. From the description of the functions to be performed by this valve gear it follows that the valve gear must be able to close the admission as well as the exhaust valves at any point of the stroke and that the action must be the same whether the engine runs over or under. These conditions are fulfilled by the long range valve gear designed by the writer, the essential parts of which are illustrated in Fig. 33. This gear admits of any adjustment of lap and lead without in any way affecting the range of release.

In order to render the working of the valve gear as clear as possible three diagrammatic views are drawn, showing separately the devices used for the different functions.

Fig. 30 shows the mechanism for operating the throttle and for regulating the expansion of air. In this A is the throttle hand lever which connects to the rocker B by means of a slotted head C.

The governor D is also connected to the rocker B in a similar manner through bell crank E and slotted head F.

The rocker B connects by rod H to the admission releasing gear, the connections being such that the point of release is advanced when the parts move in the direction of the arrow. This movement is made in opposition to the weight G on bell crank B, which weight returns the parts of the releasing gear into non-releasing position when the governor drops or throttle hand lever is in "closed" position.

The throttle I is positively operated from hand lever A by means of the rod J. K indicates the eccentric operating the wrist plate L, and M the eccentric operating the releasing cams.

By the described arrangement adjustments can be made so the throttle cannot be opened wide without applying the cut-off.

Fig. 31 is a diagram of those parts of the mechanism that determine the direction of motion of the hoist. A is the reverse lever. (In practice this is combined with the throttle lever which has a side motion operating the reversing gear, while the forward-and-back motion performs the functions of operating the throttle and cut-off.) This lever connects through the rods and levers B, B_1, B_2, B_3, B_4, to the floating lever C of the power reversing cylinder D, which through lever E operates the four-gear reversing device. This receives its motion from crank shaft G by means of parallel rods H, H_1. F_3 is the first motion shaft, from which the rotation is transmitted to the second motion shaft F by the intermediate gears F_2, F_1. The parts are shown in the position of running "under." The gears F_1 and F_2 are suspended on lever E by the link J and on the stationary shaft

bearings of F and F_3 in such a manner that when lever E is lifted from position of running under to that of running over, the rolling of the peripheries of the gear F over that of F_1, F_1 over that of F_2, and F_2 over the periphery of the stationary gear F_3, will cause the center of eccentric, which
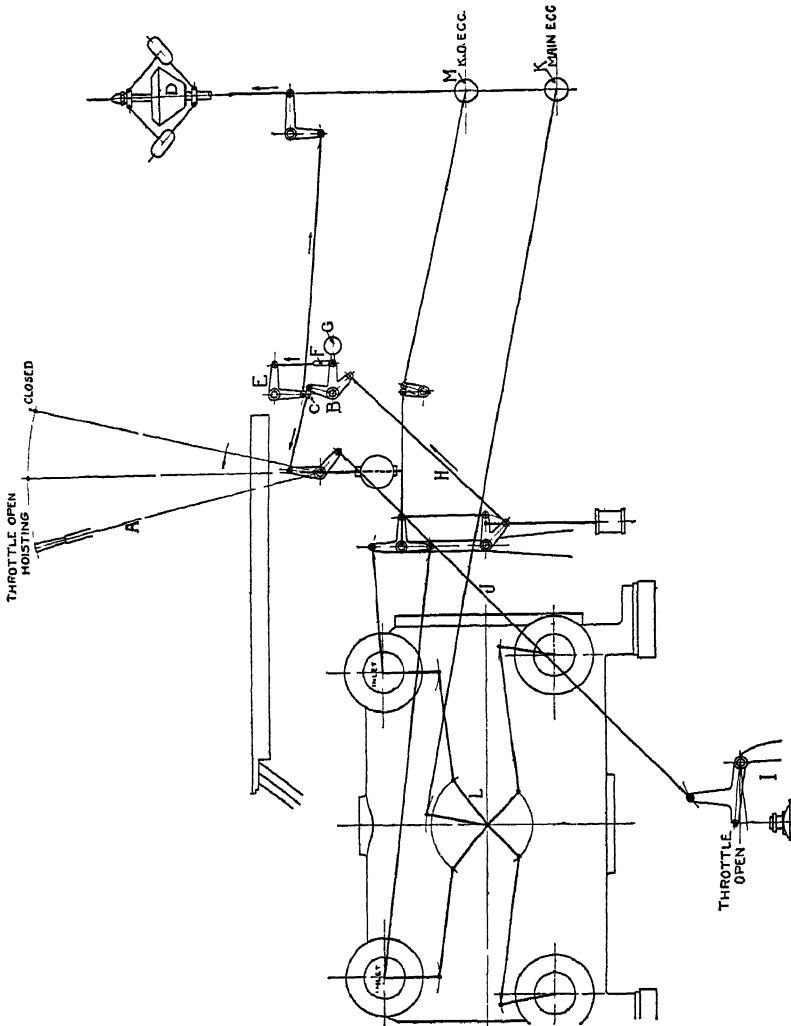
FIG. 30.—DIAGRAM OF THROTTLE, EXPANSION GEAR AND CONNECTIONS.

when running under is at X, to turn through the angle r and to assume the position Y, which is the position for running "over." Rod K connects the main eccentric to wrist plate L.

The eccentric operating the cut-off cams is mounted on the shaft M. When reversing, this eccentric has also to be turned on the shaft but through a much smaller angle. This is accomplished through the gears

F_4, F_5 and the gear F_6 mounted on the shaft M. This "secondary" reversing gear is operated from lever E through link N, the lever arm O being so proportioned as to turn the cut-off eccentric through the angle S, while the main eccentric describes the angle r. The cut-off reversing device actually used on the Butte hoists differs in details from the one described, being of a very compact design but the working principle is as described.

The mechanism used for converting the hoist from a motor to a compressor and for automatically compressing the air contained in the clearance spaces when engine is running at full speed and relieving the compression when running slow, is shown by the diagram Fig. 32. A is the regulating hand lever, B the cylinder fitted with Corliss admission valves, C, C_1 and Corliss releasing gear as described above. D, D_1 are Corliss exhaust valves, also fitted with Corliss releasing gear. E and F are the pendulum and parallelogram movements for the cut-off cams of the admission and exhaust valves respectively which receive their motion from the secondary reversible lay shaft in a manner as described above through the connections G, G_1, G_2 and G_3. H, H_1 are spring-loaded air-suction valves and I, I similar air discharge valves discharging back into the air system. J, J_1 are the bypass valves.

As has been already described, the engine is converted into a compressor by keeping the engine admission valves C, C_1 closed with reversing gear in such a position that the exhaust valves will function as they should when engine runs as a motor. Changing from motor to compressor action, therefore, is done without reversing the engine. On the forward stroke, when the exhaust valve is closed air cannot enter the cylinder through the admission valve C as it would if the valve were permitted to open and the machines acted as a motor. The result is that the suction valve H opens and air is drawn in from the atmosphere. At the end of forward stroke the exhaust valve opens and through it the air is pushed out of the cylinder up to the point at which this exhaust valve closes. The air remaining in the cylinder is then compressed and leaves cylinder through discharge valves I, I. The point at which the exhaust valve closes can be varied so that any volume from that of the whole cylinder contents to zero may be entrapped and compressed.

For this purpose the exhaust valves are fitted with a releasing gear identical to that used on the admission valves.

The regulating lever A is shown on the diagram in its neutral position, i. e., when the machine is operated as a motor. This regulating lever is connected to the separate mechanism. By the rod K it connects to the four-way rocking valve L, which is under air pressure and the exhaust of which communicates with the atmosphere. By rod N and slotted head M it connects to the three-armed rocker O, which by rod P and bell crank Q

operates the exhaust releasing gear. By the rod R bell crank R_1 and rod R_2 the regulating lever is connected to the lever S. T and T_1 are small cylinders with pistons. These pistons are forced by springs upward in the case of T and downward in case of T_1. The opposite sides of these pistons are connected by pipes to ports in the valve L whereby they can be exposed to the pressure in the air system and moved in opposition to the springs or to the atmosphere, when the springs will move them against the heads.

In the neutral position of the regulating lever the valve L is held in such a position as to communicate the cylinder T with the atmosphere. The spring in the cylinder thus holds the piston in its highest position, allowing the pin E_3 to move freely downward in the slotted head on end of piston T. In the position shown of the pin E_3 and weighted rocker E_2, which actuates the admission releasing gear through rod E_1 the admission valves admit air to the cylinders during nine-tenths of the stroke. The rocker E_2 is the same as B in Fig. 30 and is actuated by the governor so that pin E_3 may be in any position in the slot of the head on end of piston T, depending on position of governor or throttle lever. The different parts of the releasing gear on exhaust valves, rocker O, rod P, bell crank Q and pendulum F are in position to keep exhaust valves open during the entire return stroke. The rods and connections R, R_1, R_2 hold lever S in such a position that the toe on same (S_1) lifts the slotted block U_1 on end of rod U out of reach of the tooth V_1 on rocker V.

The cylinder and piston T_1 is subject to the air pressure and the piston consequently in its highest position. The extended piston rod forces the rocker V against the opposition of the spring connected therewith and keeps roller V_2 off the cam W, thus rendering the rocker V inactive. W is a revolving cam on the first motion valve gear shaft. This cam has two projections, W_1, W_2, which, when piston T_1 is down, roller V_2 held by the spring V_3 against the cam W and lever S with S_1 is in such a position as to allow the slotted block U_1 to engage the tooth V_1, give a motion to the by-pass valves J, J_1, keeping them open while the piston passes the dead center.

When the regulating lever is moved toward the left it moves the valve L so as to throw compressed air on top of cylinder T, the piston of which at once moves downward and shifts rocker E and the admission releasing gear in the direction shown by the arrow, thereby tripping the admission valves and preventing their opening during any part of the stroke. A further movement of the regulating lever in the direction of the arrow causes the inner end of slotted head M to engage the rocker O, moving it and parts connected with it in the direction of the arrows, bringing the exhaust releasing gear into action and advancing the point of release as the regulating lever is moved in the direction of the arrow. In the neutral
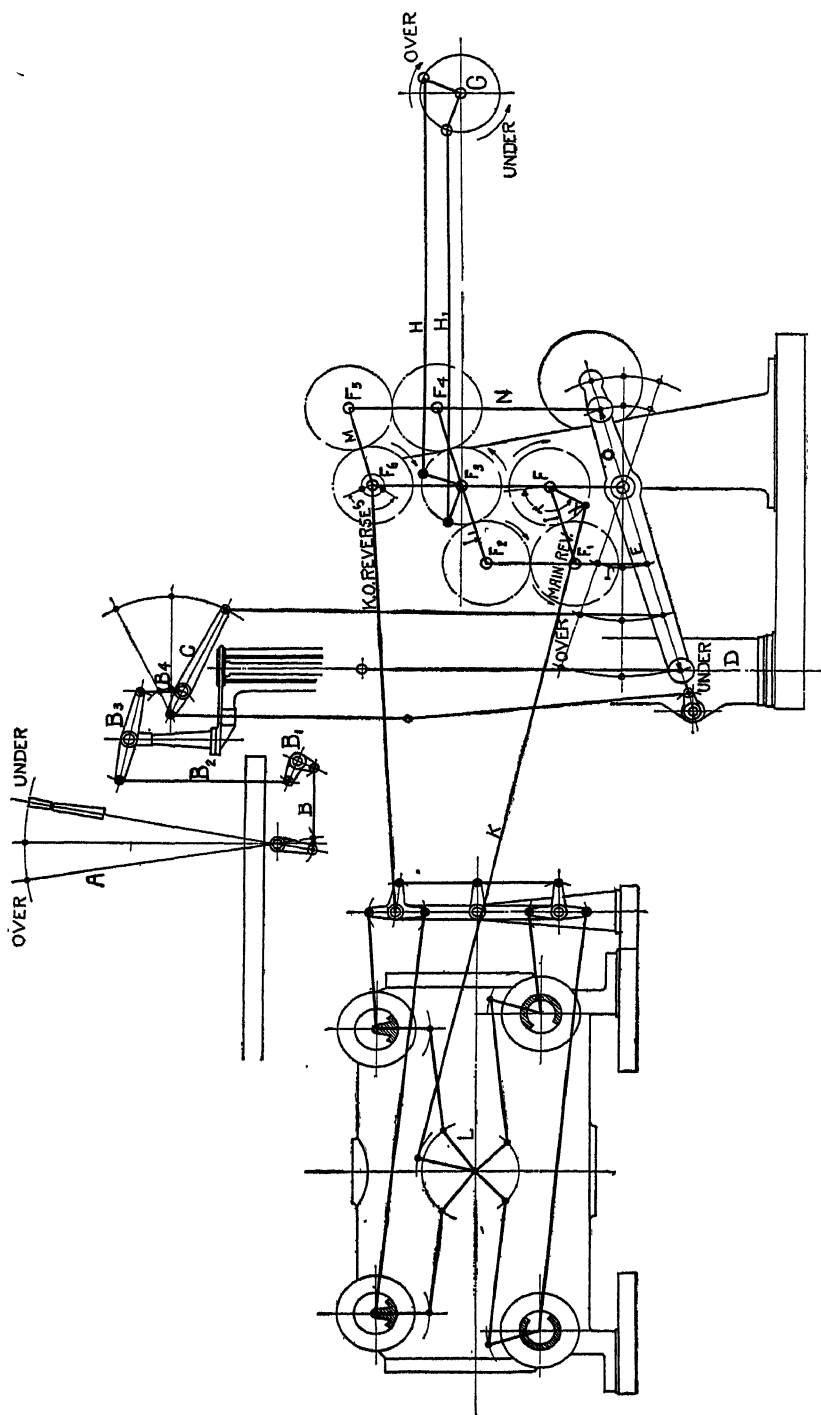
FIG. 31.—DIAGRAM OF REVERSING GEAR.
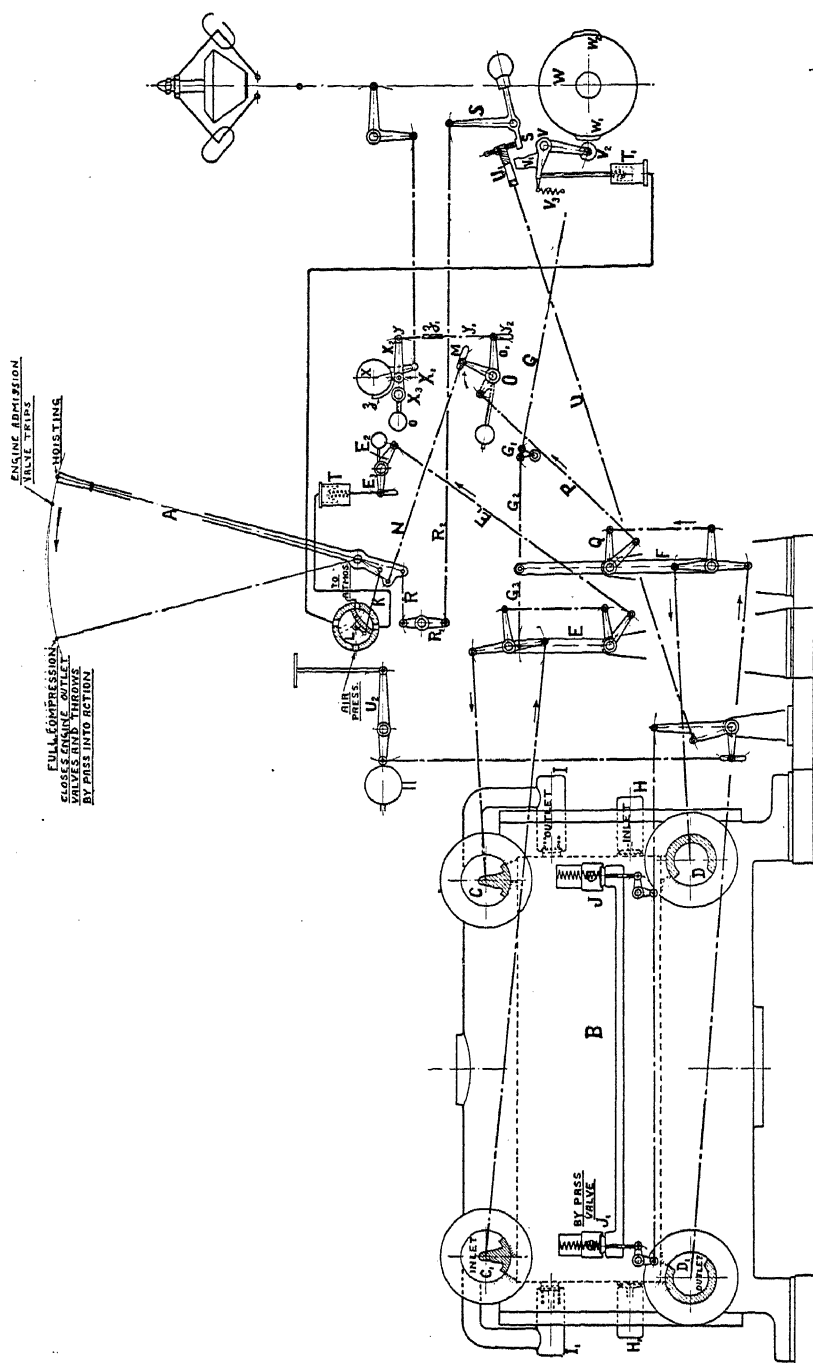
Fig. 32.—Diagram of Compression Gear and Connections.

position no air is compressed while when the regulating lever is near its extreme position a volume of air equal to the piston displacement is compressed and discharged back into the air system. We have therefore here a means to regulate the retarding force with the same precision as we regulate the development of power when the machine runs as a motor.

When the regulating lever is in its extreme left position the valve L is brought into the position as shown by the dotted lines. This puts the cylinder T_1 into communication with the atmosphere while it leaves T under air pressure. The piston T_1 is then forced down by the spring back of it and the roller V_2 on rocker V brought into contact with the revolving cam W. This causes the bypass valves J, J_1 to open simultaneously on both dead centers of the engine and to close as soon as the air in the clearance space back of the pistons has passed over to the other side of said pistons and an equilibrium of pressure is established between the two sides. The initial air pressure will by this process increase from atmospheric pressure to about 3 lb. above atmosphere.

When this air is compressed a considerably higher average pressure is obtained than that resulting from compressing the full piston displacement of free air, particularly as the work of expansion, which is positive, disappears. Fig. 28k shows the indicator card of this intensified compression as compared with the regular full power compression card.

With the intensified compression the heaviest downgoing loads can be quickly retarded and stopped and the device as described for this purpose is an important improvement in the operating mechanism of an air hoist.

The bypass valves can be operated by a foot lever in the regular manner for landing the skips and maneuvering. These valves are at such times not actuated by the cam W.

For the purpose of recompressing air to fill the clearance spaces with air of initial pressure when the engine is up to full speed, the following device is used in connection with the exhaust releasing gear: The governor actuates a cam X engaging a roller X_1 mounted on a lever X_2 hung on a fixed pivot X_3. The pin y on end of this lever connects with rod y_1 and slotted head y_2 to arm O_1 on rocker O. When now the hoist is up to speed and governor commences to rise, the projection z on cam x is brought in contact with roller X_1 and as the governor lifts will move the lever X_2, rod y_1 with slotted head y_2 downward, thereby moving rocker O and the parts of exhaust valve releasing gear connected therewith in the direction of the arrows, thereby advancing the point of closure of the exhaust valves an amount proportional to the radial dimensions of the projection z. This dimension is such that the advanced closure of the exhaust valves entraps just enough air in the cylinders so that when it is compressed the final pressure is 90 lb. per sq. in. By means of the right and left hand screw adjustment z_1 the point of release can be accurately set by the aid

of the indicator. As long as the governor stands a small distance over its collar and the engine is up to speed the compression as described takes place. This not only entirely eliminates the clearance loss but produces smooth running of the engine.

When the speed is dropped and the governor returns to its collar the cam X and projection z of same are automatically moved out of reach of roller X_1. The rocker O and parts connected to same are thereby caused
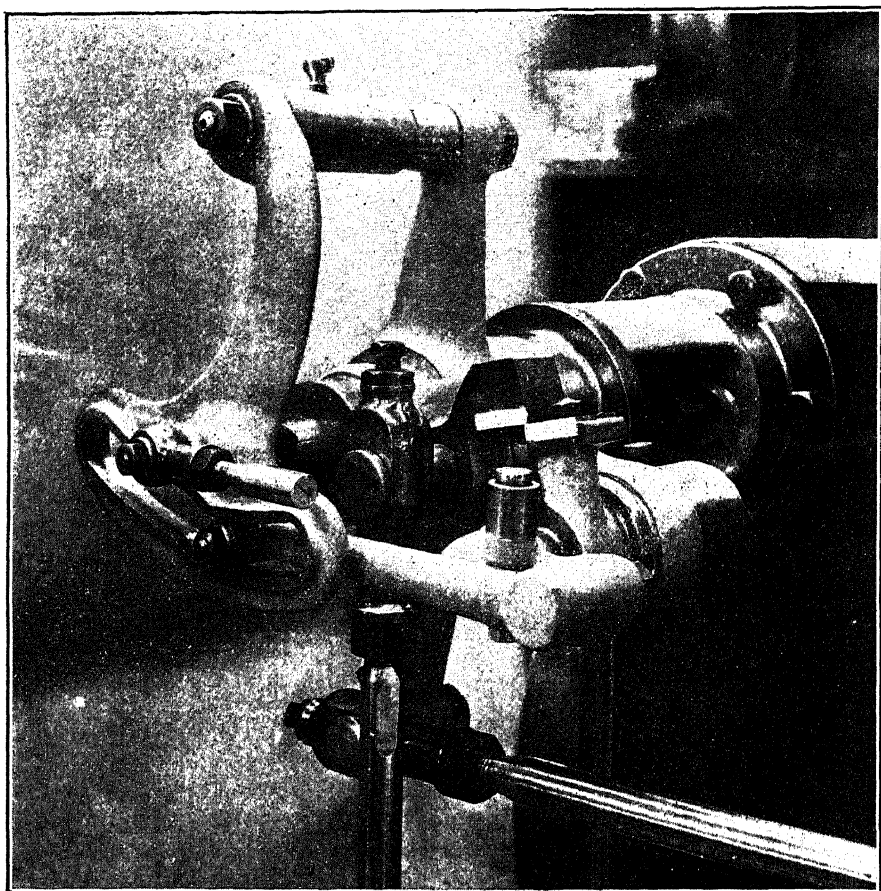
FIG. 33.—FULL STROKE VALVE GEAR.

to return into the position of non-release of the exhaust valve gear. The compression which, if present under low speed, would seriously interfere with the maneuvering of the hoist, now disappears.

The continuous indicator diagrams Figs. 34 to 38 were all taken from different air operated hoists in the Butte camp, fitted with the above de-scribed valve gear, and show how well it performs the different functions.

SCALE 1 INCH = 240 LBS PER SQ IN

19.5 LBS. — AVERAGE M.E.P.

M.E.P.

REVOLUTIONS
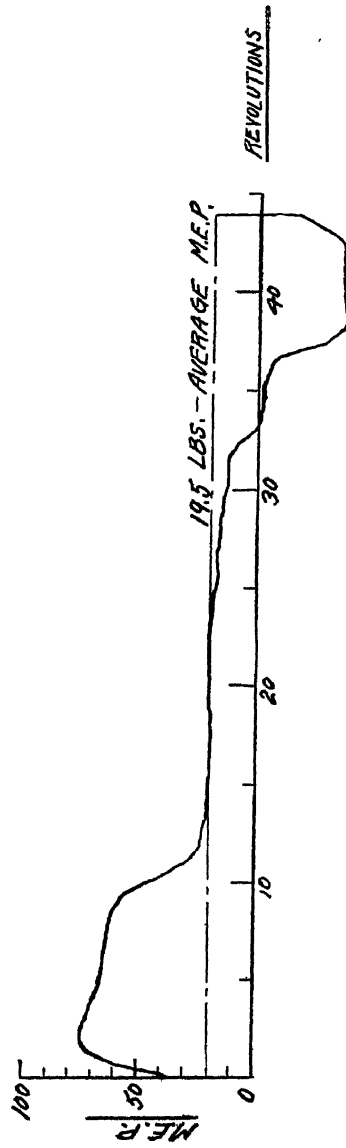
FIG. 34.—HOISTING CARD TAKEN AT THE LEONARD MINE FROM THE RIGHT-HAND HEAD END OF A 34-34 BY 72 AIR HOIST, 12-FT. DRUM, RUNNING BALANCED. SKIP OF ORE HOISTED 1,800 FT. IN 45 SECONDS.

SCALE 1 INCH = 240 LBS PER SQ.IN

100
50
MEF

4.7 LBS = AVERAGE M.E.P.

10 20 30 40 50 60 70 80 90
REVOLUTIONS
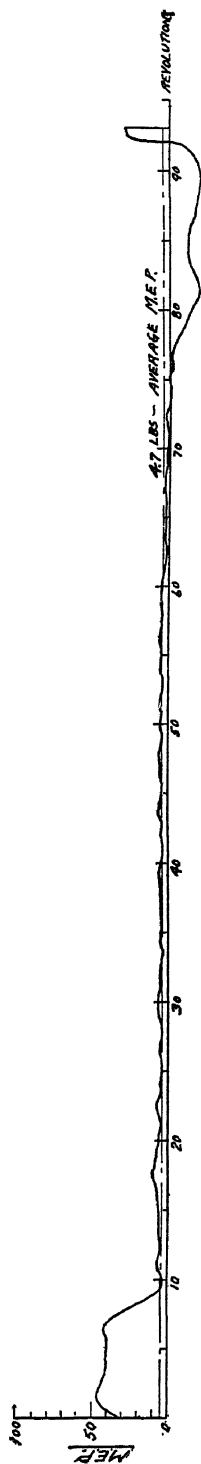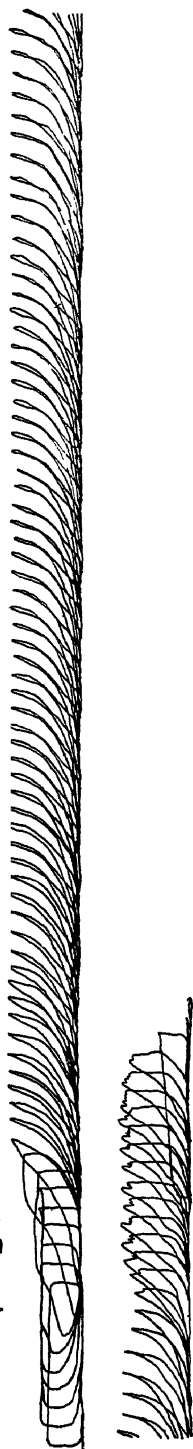
FIG. 35.—HOISTING CARD TAKEN AT THE MODOC MINE FROM THE LEFT-HAND HEAD END OF A 28–28 BY 48 AIR HOIST, 6-FT. DRUM, RUNNING BALANCED. TWO DECKS OF CAGES AND TWO CARS OF WASTE HOISTED 2,000 FT. IN 55 SECONDS.

SCALE 1 INCH = 240 LBS. PER SQ IN

18.26 LBS. - AVERAGE M.E.P.
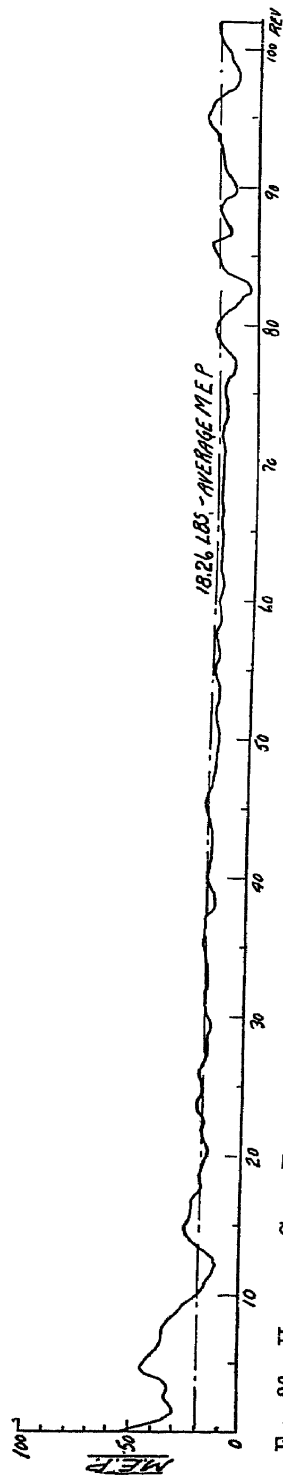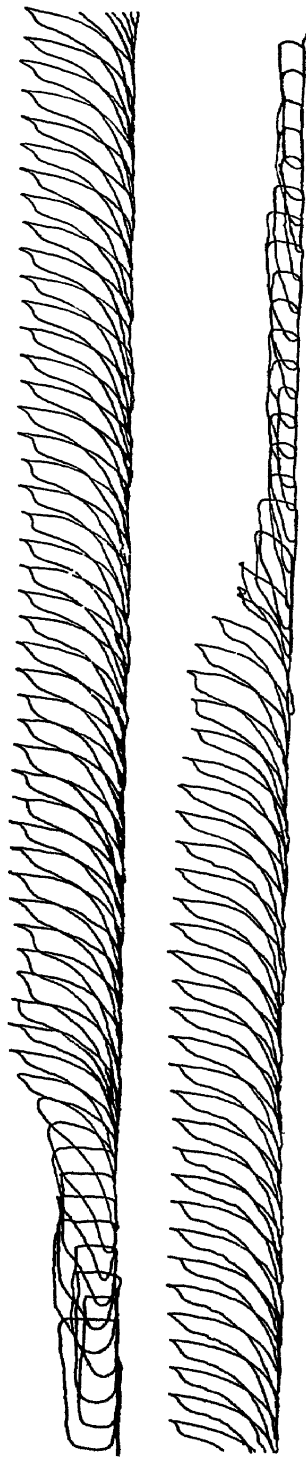
FIG. 36.—HOISTING CARD TAKEN AT THE TRAMWAY MINE FROM THE LEFT-HAND CRANK END OF A 28-28 BY 48 AUXILIARY AIR HOIST, 6-FT. DRUM, AIR PRESSURE 92 LB. PER SQUARE INCH, RUNNING UNBALANCED. THREE CAGES AND THREE EMPTY CARS HOISTED 2,000 FT. IN 60 SECONDS.

Fig. 34 is taken from the main hoist at Leonard mine when hoisting a balanced load of ore from the 1,800 ft. level. The first five strokes require full filling of the cylinders. Five more strokes are performed with reduced filling but at the 11th stroke the clearance compression gear comes into action. The filling is gradually reduced to zero and at the six strokes at the end of the run air is compressed back into the system.

This hoist has cylinders 34 x 72 in. and 12 ft. drums carrying 1½ in. ropes.

Fig. 35 is from the new 28 x 48 in. double drum hoist at the Modoc mine. The drums here are 6 ft. diameter and carry 1 in. ropes. This card shows how efficiently the described valve gear works with very light loads. As there is no clearance loss during the greater part of the run the air consumption is very small. The card shows that for retarding the hoist air was compressed back into the system during fourteen strokes.

Figs. 36 and 37 are indicator cards from the single drum auxiliary hoist at the Tramway mine. This hoist has cylinders 28 x 48 in. and the drum is 6 ft. diameter. The work of this hoist is mostly negative as it is used for lowering waste rock into the mine to fill out empty stopes. It runs out of balance.

Card Fig. 36 is taken when the empty cages are hoisted while card Fig. 37 is taken when the cages loaded with 3 cars of waste rock were lowered into the mine. The machine is run as a compressor when this work is done.

Card Fig. 38 is from the main hoist at the High Ore mine. This hoist is fitted with reels for flat ropes. The card is taken when hoisting from a depth of 2,800 ft. in balance. Here the air is shut off after a little over two-thirds of the run is completed. The balance of the run is made with the momentum of the moving masses as a driving force which, at the end of the run, just balances the friction and static moment of the load.

The mean effective pressures of the individual cards of these continuous diagrams are plotted below the diagrams. The average values of all the mean effective pressures are also given.

The installation of this plant was commenced in the latter part of 1910, when the hoists at Mountain View, High Ore and Diamond were reconstructed and three compressors installed. The Mountain View hoist was started on air in the early spring of 1911 and the other two soon thereafter. The performance of these three hoists was regarded so successful that six more hoists were changed over and twelve new auxiliary hoists ordered all of which were designed on the same principles as the original three hoists.

The compressor plant was also extended. It now contains six compressors. Two more compressors are being built. About three-fourths of the air from this plant is used for operating the hoists and the balance is used to help out the rock drill system.

SCALE 1 INCH = 320 LBS PER SQ.IN

-37.2 LBS.-AVERAGE MEP

M.E.P.

REV
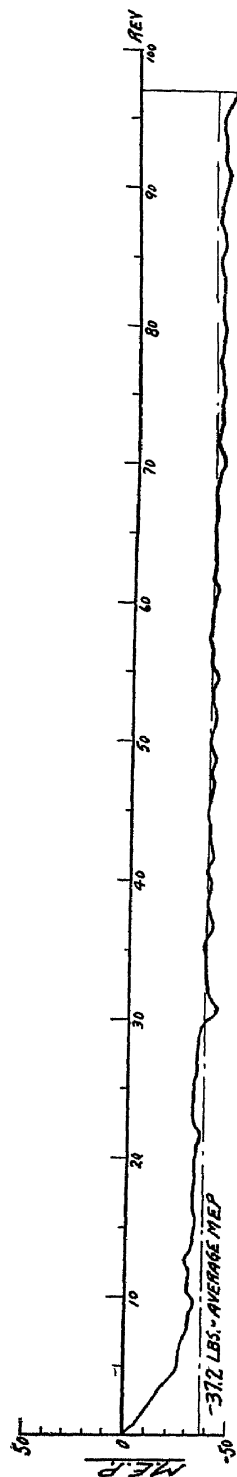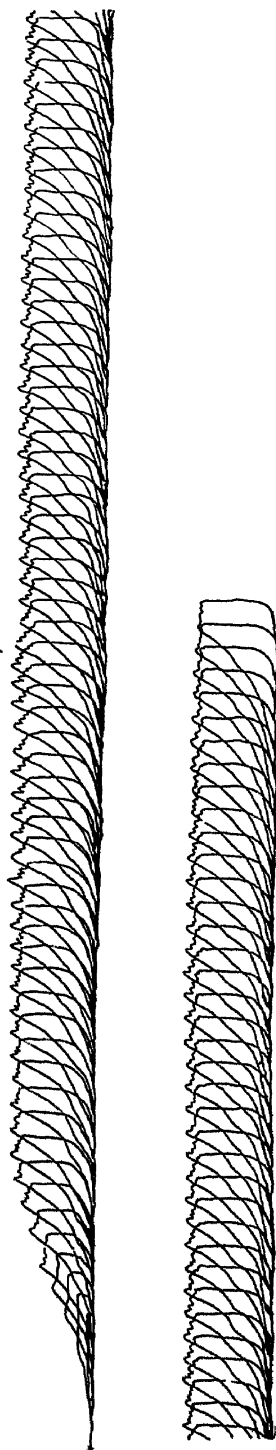
FIG. 37.—LOWERING CARD TAKEN AT THE TRAMWAY MINE FROM THE RIGHT-HAND HEAD END OF A 28–28 BY 48 AUXILIARY AIR HOIST, 6-FT. DRUM. THREE DECKS OF CAGES AND THREE CARS OF WASTE LOWERED 2,000 FT. IN 70 SECONDS.
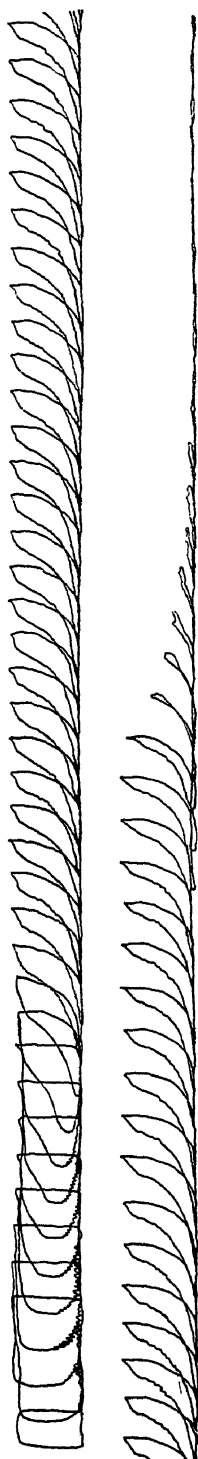
SCALE 1 INCH = 240 LBS. PER SQ IN
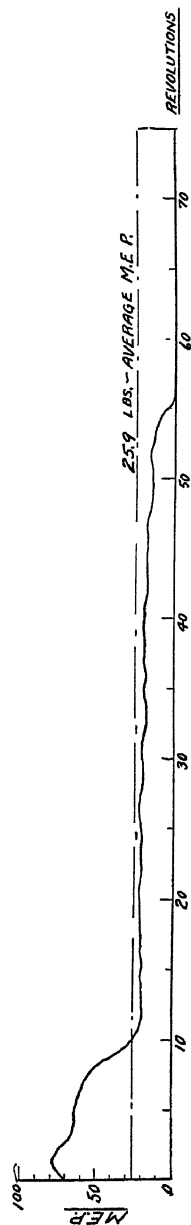
FIG. 38.—HOISTING CARD TAKEN AT THE HIGH ORE MINE FROM THE LEFT-HAND CRANK END OF A 34-34 BY 72 AIR HOIST RUNNING BALANCED. SKIP OF ORE HOISTED 2,800 FT. IN 60 SECONDS.

It was originally contemplated to regulate the hoisting so that the active periods of the different mines would not occur at the same time. So far it has not been found practical to do this to any great extent, the result being that at times the storage system is overtaxed.

The advantages of this adopted system over any in which electric motors are directly applied to the hoists are the following:

(1) The existing hoisting engines could be retained and needed comparatively few changes.

(2) They could be operated by steam in case of a serious accident to the power system.

(3) Enough energy is always available in the storage plant to operate the hoists for a short time in case of a short failure of the electric power due to thunder storms, etc.

(4) The capacity of the hoists can be increased, by increasing the hoisting speeds of the loads or the hoisting depth can be increased without changing the motor on the hoist. The compressor capacity only needs to be increased.

(5) A considerable part of the energy liberated in retarding the hoists at the end of a run or when loads are lowered into the mine is returned to the power system and is held in storage without loss for an extended period.

(6) The system can be started with its stored energy without subjecting the electric apparatus to excessive peak loads as is the case when a flywheel of an Ilgner transformer is started.

(7) By this system of compressing, storing and using the air in the hoist cylinders, peak loads on the electric apparatus are absolutely avoided. No matter how the load on the hoist motors may fluctuate, the power to operate the electric motors in the compressor plant is absolutely constant.

(8) Compared with electric hoists the air system has the following electrical advantage: With the air hoist, the electric prime movers are synchronous motors which can be operated with a leading power factor and therefore raise the power factor of the entire electrical system, while with an electric hoist, the prime mover is an induction motor which would have a very low power factor due to the intermittent loads it would have to carry to meet the Butte conditions of hoisting and hence would decrease the power factor of the entire electric system.

As compressed air is used for operating the rock drills it seems consistent and natural, when electric power is used for compressing this air, to enlarge the compressing plant so that the hoists also are operated by air. In most cases of ore mining it will be found that the increase in compressor capacity necessary for hoisting with air is not very large. If proper air storage is provided, the result will be a simpler, cheaper, safer and more economical plant than if the hoists were directly driven by electric motors

and motor generators with heavy and fast rotating flywheels were used. The high peak loads resulting from starting these flywheels and from the irregularity of hoisting will be avoided and the electric load will be perfectly constant.

The plant has been a success and the writer believes that it has fulfilled the expectations of the owners. The mechanism used for performing the different functions in both the compressors and the hoists were new, and applied in practice for the first time in the plant. Since the machinery was started there have been no interruptions in its work and very little trouble.

In a new plant operated with compressed air many improvements could be made that were not possible in the Butte plant. Some of these improvements would be of very radical nature and would result in obtaining very high efficiencies without in any way impairing the reliability of the machinery or ease of operation, which, after all, is the most important thing in a mine hoist.

The Coal Fields of Montana.*

BY EUGENE STEBINGER,† WASHINGTON, D. C.

CONTENTS.

INTRODUCTION.

A LARGE number of papers and reports dealing with the coal fields of Montana have been published ‡ during the last 30 years, but the information is much scattered, appearing in many technical and scientific journals and in government reports, in many cases inaccessible to those most interested. In recent years, especially since 1905, our knowledge of the geology and coal contents of the important fields of the State has been much increased. Fairly specific and detailed information is at hand on all of the principal coal-bearing areas. The fields for which data are not at hand are small and inaccessible, and their omission will not materially affect an estimate of the coal resources of the State, viewed as a whole.

The paper here presented is a summary of existing information, all sources having been freely utilized in its preparation. The viewpoint adopted is that of the geologist, rather than that of the engineer or colliery operator, because by far the greater portion of the coal areas present have had only the most meager development, making information that will aid in prospecting and opening the at present little used coal lands of paramount importance to those interested in the coal fields of the State.

POSITION OF MONTANA IN THE WESTERN COAL PROVINCES.

Conditions favorable to the accumulation of coal existed during Cretaceous and early Tertiary times in the United States between the 100th and 115th meridians, resulting in the accumulation of an extensive series of coal formations occupying a large part of the region between those meridians and extending from north to south entirely across the country and reaching into Canada and Mexico. The western half of this immense area of coal-bearing formations, soon after being deposited, was subject to the series of mountain-building movements which produced our present Rocky Mountain system. Because of these mountain-making disturbances and subsequent erosion, the coal formations originally more or less continuous in this area were isolated into a large number of detached basins. The area occupied by these basins has been called the Rocky Mountain coal province.

On the other hand, the coal-bearing formations laid down in the eastern one-half of the area lying between the 100th and 115th meridians were not disturbed by the orogenic movements producing the Rocky mountains and, where not covered by later deposits, are

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‡ See list of literature at end of paper.   Reference numbers in the text refer to this list.

now found lying almost horizontal over large areas in the northern Great Plains region and make up an immense continuous coal-bearing area that has been called the Northern Great Plains coal province. Montana contains a portion of each of these large coal provinces, the western part of the State with numerous isolated coal basins making up a portion of the Rocky Mountain province, and the eastern part of the State making up a large unit area of the Northern Great Plains province. (See map, Pl. I.)

MONTANA'S TOTAL COAL TONNAGE.

Sufficient detailed geologic work has been done on the western coal fields during the last six or eight years to allow estimates to be made of the total coal tonnage in the important coal-bearing States. These results are shown graphically in Fig. 1, the fuels being classified into three grades, bituminous coal, sub-bituminous coal, and lignite. Montana, with a total of 380 billion short tons of bituminous coal, sub-bituminous coal, and lignite, ranks third in total coal tonnage, compared with the other States in the West. These results are especially interesting in view of the fact that earlier workers,[32, 41] basing their estimates on the probable areas underlain by coal-bearing formations in each of these States, place Montana first in a list of the coal-bearing States. North Dakota, with its immense tonnage of lignite, and Wyoming, with almost as great a tonnage of much higher grade fuel, each contain almost double the Montana tonnage, of which one-half is made up of lignite of the same grade as that found in North Dakota. If the grade of coal present be taken into consideration in the comparison of the coal resources of each of these States, Montana, with over one-half of its tonnage made up of lignite and only a very small proportion of the remainder consisting of true bituminous coal, should certainly rank below Colorado and Utah, both of which contain a very large tonnage of bituminous coal, and probably also below New Mexico, which contains twice Montana's tonnage of bituminous coal and a slightly larger tonnage of sub-bituminous coal. (See Fig. 1.)

THE COAL-BEARING FORMATIONS.

General Statement.

The coal-bearing formations of Montana range in age from Lower Cretaceous to mid-Tertiary, the oldest coal-bearing rocks being those of the Kootenai formation, which contains the coal horizon being mined in the Lewistown and Great Falls fields, and the youngest being the Tertiary lake beds, at present mined in the Drummond and Missoula fields.

PLATE 1.—MAP OF MONTANA SHOWING THE COAL AND LIGNITE FIELDS.

Lignite.

Probable lignite area.

Sub-bituminous coal.

Probable area of sub-bituminous coal.

Bituminous coal.

Probable area of bituminous coal.

Anthracite or semi-anthracite, with probably a large percentage of bituminous coal.
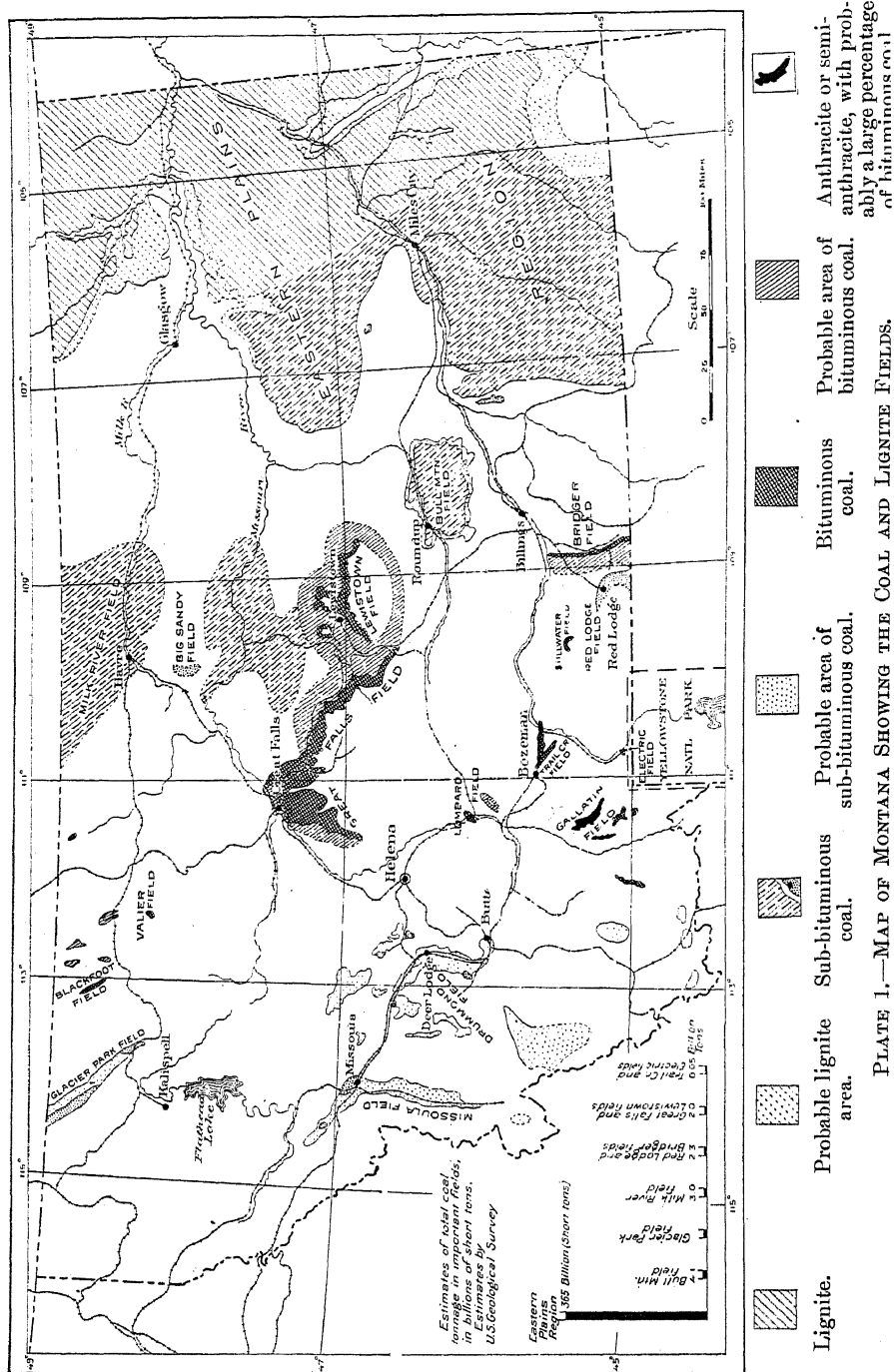
The complete sequence of the coal-bearing formations in this State is shown in the generalized columnar section of the Cretaceous and Lower Tertiary rocks presented in Fig. 2. All of these formations are not present within any one field in the State, but the superposition of the various rock groups is definitely known because of the extensive geologic work that has been accomplished. Seven of the
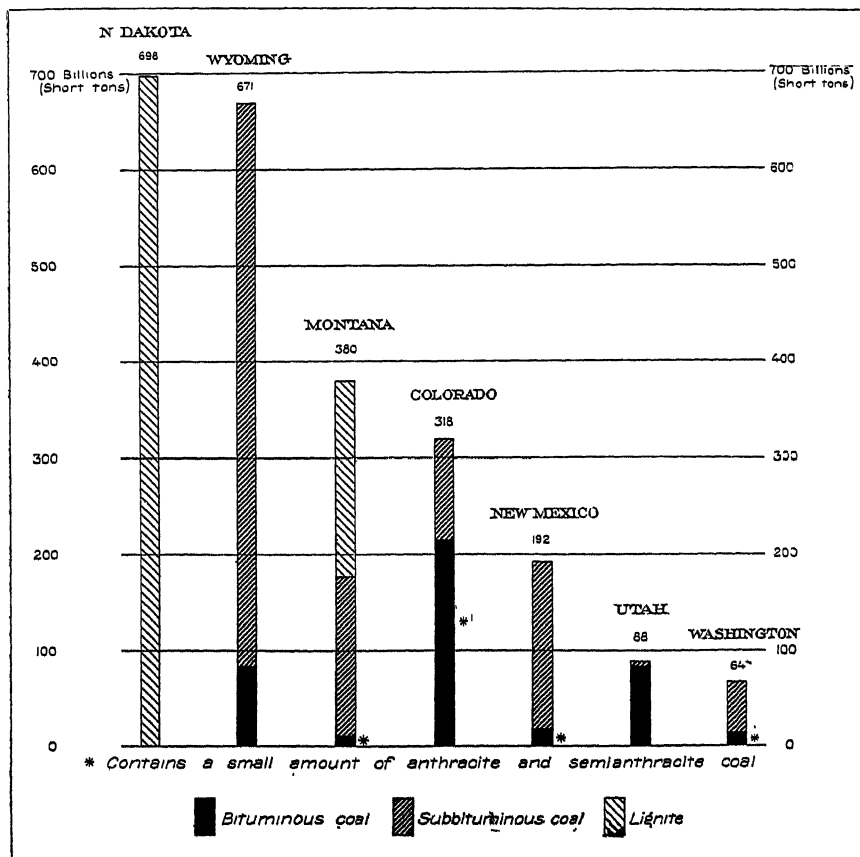
FIG. 1.—MONTANA TONNAGE OF ALL GRADES OF COAL AND LIGNITE COMPARED WITH THE TOTALS FOR OTHER WESTERN STATES. THE TOTALS ARE FOR WORKABLE COALS WITHIN 3,000 FT. OF THE SURFACE.

(From estimates by the Fuel Section of the U. S. Geological Survey, under the direction of M. R. Campbell.)

ten formations listed in this section, beginning with the Kootenai and ending with the Tertiary lake beds, are known to contain coal of commercial value in some parts of the State, but, on the other hand, not one of these seven contains workable coal in every locality in
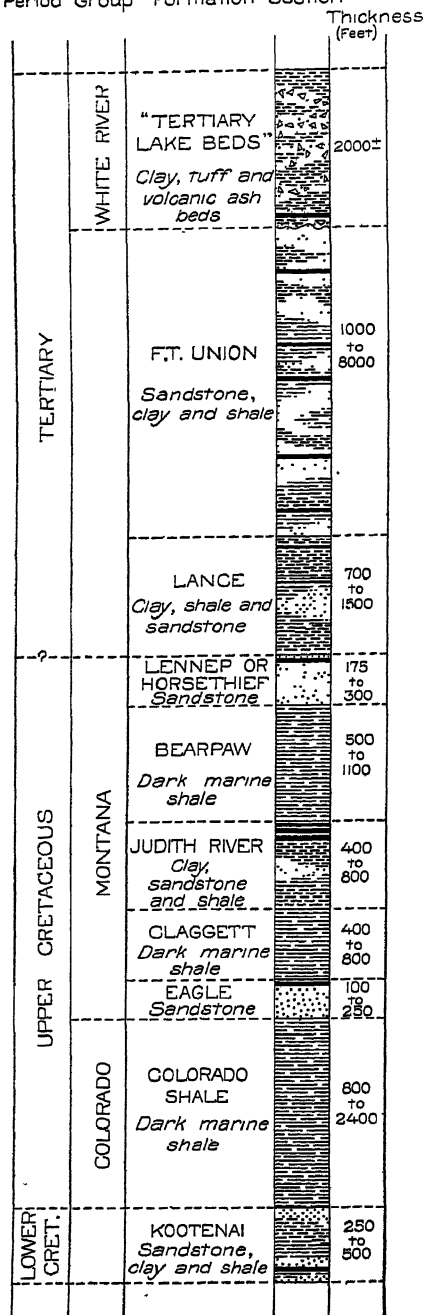
Coal in the Drummond and Missoula fields, and in the other areas of Tertiary Lake beds in the mountains in the west half of the State occurs at this horizon.

Coals of the Red Lodge, Bull Mountain and Big Sandy fields and nearly all of the coals in the Eastern Plains region occur in this formation.

A few thin and usually dirty coal beds occur at this horizon in the Bull Mountain field and in the Eastern Plains region.

Coal of the Blackfoot field, Teton county, occurs at this horizon.

Horizon of the coals of the Milk River field.

Horizon of the coal in the Bridger field, the Valier field and the approximate position of the coals mined in the Electric and Trail Creek fields.

Coal horizon of the Great Falls and Lewistown fields.

FIG. 2.—GENERALIZED COLUMNAR SECTION OF THE CRETACEOUS AND TERTIARY FORMATIONS IN MONTANA, SHOWING THE STRATIGRAPHIC POSITIONS OF THE COAL HORIZONS IN THE VARIOUS COAL FIELDS.

which it is found. This uncertainty in the continuity of workable coal horizons is generally characteristic of a coal-bearing formation.

The formations that are not coal-bearing are the Colorado shale, the Claggett formation, and the Bearpaw shale, each of which is a dark gray marine shale, very similar lithologically. They are mainly of a marine character, this being proved by the abundance of ammonites, baculites, and other marine invertebrate fossils found in them. These marine shales mark three separate stages, separated by intervals in which the Eagle sandstone and Judith River formations, each containing a few coal beds, were deposited, during which all or a large part of the State was covered by the comparatively shallow marine waters of the ancient Cretaceous sea.

The principal lithologic characteristics and other stratigraphic details of the various coal-bearing formations, beginning with the oldest and crossing upward through the geologic time scale, are given below.

Kootenai Formation,[3, 4, 14, 15, 22, 23.]

The Kootenai formation is exposed in many parts of western Montana and is especially important in the Lewistown and Great Falls fields, because it contains the coal mined in those areas. Lying stratigraphically below a considerable thickness of Colorado shale, it is readily distinguished as a formation by the bright maroon-colored shales occurring throughout it and also by the prominent and frequently cross-bedded sandstones occurring in the lower part. The formation varies from 250 to 500 ft. in thickness in central Montana, while in the northwestern part of the State it is probably as much as 2,000 ft. thick. The typical development of this formation in the region in which it is coal-bearing is illustrated by the following section :

Generalized Section of the Kootenai Rocks in the Lewistown Coal Field.

		Feet.
(1)	Shale, maroon, argillaceous, ...	200
(2)	Sandstone, grayish, coarse-grained, cross-bedded, ...	8
(3)	Shale, maroon, argillaceous, ...	60
(4)	Sandstone, gray, coarse-grained, cross-bedded, ...	25
(5)	Shale, maroon, argillaceous, ...	72
(6)	Sandstone, weathering soft gray, massive, coarse-grained to pebbly, . .	50
(7)	Coal and coaly shale, ...	10
(8)	Shale and sandstone alternating ; the shale is sandy and the sandstone thinly bedded, ...	87

The Lower Cretaceous age of the group of rocks making up this formation has been definitely determined by means of the abundant and well-preserved fossil-plants occurring throughout the strata, which

made it possible to correlate them with the Kootenai formation, as originally described by Dawson in Alberta. The rocks are all either of terrestrial or fresh-water origin, as is indicated both by the flora and by the few invertebrate fossils found.

Wherever the Kootenai formation contains workable coal in Montana the coal is found in a single bed occurring at one horizon lying between 60 and 90 ft. above the base of the formation. Immediately above this coal horizon there is generally found a massive, coarse-grained to pebbly sandstone, which, where the formation dips to an appreciable degree, weathers out as a prominent strike ledge, thus affording an excellent horizon marker to one prospecting for the coal.

Eagle Sandstone,[6, 7, 25, 42, 44.]

The coal in the Bridger field, in Carbon county, and that in the small area designated as the Valier field, in Teton county, occur in the Eagle sandstone. In the Electric and Trail Creek fields the various formations belonging to the Montana group of the Upper Cretaceous cannot be sharply differentiated, but the coal being mined there is found at a position approximating that of the horizon in the upper part of the Eagle sandstone.

Where it is typically developed, as on the Missouri river near the mouth of Eagle creek, where it forms the prominent landmark known as the " Stone Walls of the Missouri," the formation is characterized by a massive gray to whitish gray sandstone. Its position above the dark bluish gray Colorado shale and the prominence of its main sandstone member make the formation easily recognizable, although the fossils found are of little value for purposes of correlation. In the Valier field the coal in the Eagle lies at the top of the formation and is present in a single workable bed, while in the Bridger field it occurs in three beds, separated by massive sandstone members from 35 to 75 ft. thick.

The Eagle sandstone has been recognized at many other localities over a wide area in the western part of the State, notably (1) in the upper part of the Musselshell valley north of the Crazy mountains,[42] (2) in the lower Musselshell valley north of the Bull mountains, and (3) in Teton county over an area extending from Chouteau northward to the international boundary, and, although thin coals are occasionally found in these localities, the presence of coals of commercial value is very doubtful.

Judith River Formation,[25, 26, 42.]

The Judith River formation is coal-bearing to a commercial extent in two localities in Montana, the first in the Milk River field in Chou-

teau county in the northern part of the State, and the second just to the south of the above area along the Missouri river in the vicinity of the Judith basin. In its typical development, the Judith river is almost entirely a fresh-water formation. It overlies the marine Claggett shales conformably and is composed of alternating light-colored beds of sandstone and shale, having an average thickness of 380 ft. A few brackish water beds occur near the top of the formation in a transition zone grading into the marine shales of the Bearpaw lying above. There is no persistent member in the entire formation which can be followed any great distance, the occasional beds of massive sandstone present grading rapidly along the outcrops into soft shale or friable sandstone. In the same manner a coal bed of workable thickness which may be present at one locality may pinch out altogether and be replaced by carbonaceous shale at the next exposure half a mile away.

All of the workable coals found in the Judith River formation occur within the upper 150 ft. As many as four beds have been found at a single locality at intervals of from 10 to 30 ft. apart, but usually only one is found. The beds are very lenticular and show considerable variation in thickness when traced on the outcrop. The Judith River formation has been studied over nearly the entire length of the Musselshell valley in the central part of the State, but was not found to contain coals of importance in any of this area.

Horsethief or Lennep Sandstone,[42.]

Overlying the Bearpaw shale wherever the beds are not eroded away, there is generally found a sandy formation more or less similar to the Eagle sandstone, and, like it, containing a few thin coals in its upper portion over widely scattered localities. The most important coal occurrence associated with this sandstone is in the Blackfoot field in Teton county in the northern part of the State, where coal is found outcropping at intervals, along a tract about 15 miles in extent. The formation is made up almost entirely of a massive gray sandstone, varying from 175 to 300 ft. in thickness and containing a single non-persistent coal horizon at the top. This sandstone formation also outcrops in the part of the Musselshell valley lying immediately north of the Crazy mountains, where it is known as the Lennep sandstone. It is not coal-bearing in this latter area.

Lance Formation,[5, 8, 16, 21, 37, 54.]

The Lance formation is of small importance as a coal-bearing formation in Montana. It forms the western border of the large

area making up the lignite and sub-bituminous coal region in the eastern part of the State. A large part of this border is mapped as being doubtfully coal-bearing because of the very uncertain nature of the Lance coals contained in it. The exact age of the Lance has been in dispute for many years. The fossil flora found in the rocks indicates a Tertiary age for this formation, while the invertebrate fauna is closely allied to similar forms found in the Cretaceous. The formation consists of a variable thickness of lenticular beds of soft clays, shales, and sandstones, and is generally distinguishable from the formations lying above and below it by its sombre-colored hues, in recognition of which the formation has often been termed "Sombre Beds."[8] The few coals contained in it are found in the upper part of the formation and are thin, lenticular, and generally very dirty.

Fort Union Formation,[1, 2, 5, 8, 9, 12, 16, 37, 39, 40, 53, 54.]

About 90 per cent. of the total tonnage of lignite and coal found in Montana occurs in the Fort Union formation, making it by far the most important of the rock groups under consideration. It contains practically all of the coal in the Bull Mountain field, the Red Lodge field, the Big Sandy field, Chouteau county, and in the immense coal area in the Eastern Plains region. The only known locality where rocks occupying this position in the stratigraphic column in Montana do not contain workable coals is in Teton county in the northwest part of the State, where only a few very thin and scattered coals occur.

The Fort Union formation consists of a variable thickness of yellowish-grayish sands and sandstones, often in quite massive beds, interbedded with gray clays and coals. The sands and sandstones predominate. In the eastern part of the State it is not much over 1,000 ft. in thickness, but in going westward toward the mountains it thickens until, in the Red Lodge field, it is 8,500 ft. thick. A very complete and well-preserved fossil flora is found in these rocks, which places their age as basal Tertiary. Leaves from broad-leaved trees, very similar to present existing species, are the most common forms met with.

The number of coal beds and the total amount of coal found in this formation are phenomenal. In the Bull Mountain field 24 beds were found at as many different horizons in a total thickness of 1,650 ft. of strata. The general habit of the beds is lenticular, although several of the coals are notable exceptions to this rule, and it has been possible to trace their outcrop for many miles. In the Eastern Plains region the coal beds are equally as numerous and, on the

average, rather thicker than in the Bull Mountain field. In Dawson county, on the Yellowstone river, a section across 980 ft. of this formation contains 11 lignite beds totaling 49 ft. in thickness, giving a ratio of 1 ft. of lignite for every 20 ft. of rocks. This is about double the amount of coal per unit of formation found in the eastern Coal Measures, and justly entitles the Fort Union to be termed a "coal age," as well as a Carboniferous period in Palæozoic time.

Tertiary Lake Beds.

The coals in the Missoula and Drummond fields, and numerous smaller areas in the mountains of western Montana, occur in the Tertiary lake beds, which are the youngest coal-bearing rocks in the State. They have been determined to be approximately Oligocene or mid-Tertiary in age by means of the vertebrate fossils contained in them or in immediately associated rocks. They are made up mainly of massive slightly indurated tuff composed in part of clay, the prevailing colors being light shades of cream and gray. They also contain occasional beds of fairly pure volcanic ash, and also impure fresh-water limestones. The coal found in these beds seems to occur only at horizons near the base of the formation.

DISTRIBUTION OF THE COAL FIELDS.

The widespread distribution of lands containing coal in Montana is strikingly shown by the map of the coal fields of the State (Pl. I.). Coal fields of greater or less size are found in every part of the State, so that there are few points in it that are more than 75 miles from an area containing workable coal. The largest single area present is the lignite and sub-bituminous coal region in the eastern part of the State, covering a large part of Custer, Dawson, and Valley counties, and extending entirely across the State in a north-south direction, with a maximum width of 160 miles. This immense area is estimated to contain 365 of the total 380 billion short tons of coal and lignite occurring in Montana. A graphic representation of the tonnage present in this region compared with other important fields in the State is shown in the diagram on Plate I. The next important field in point of contained tonnage is the Bull Mountain field, with the estimated total of a little over four billion tons of sub-bituminous coal. None of the remaining contains more than three billion tons of workable coal. Because of their proximity to means of transportation, the Bridger, Red Lodge, and Trail Creek fields, which are comparatively unimportant when the total coal tonnage of the State is considered, have, up to the present time, furnished the greater part of the coal

mined in Montana. The immense quantities of fuel found elsewhere remain practically undeveloped.

The central part of the State, going from south to north, contains, first, near the southern border, a group of comparatively small fields including the Bridger, Red Lodge, Stillwater, Trail Creek, Electric, Gallatin, and Lombard fields. Farther north, in the middle part of the State, lie the Bull Mountain, Lewistown, and Great Falls fields, all of which are much larger in area than the fields lying farther south, while still farther north lies the Milk River field, bordering on the international boundary. The Big Sandy field, and an unnamed area occupying the region along the Missouri river, lie between the Lewistown and Milk River fields.

The western part of Montana, which lies mainly within the Rocky mountains, is dotted with numerous small isolated coal fields which have undergone little or no development and concerning which very little detailed information has been published. The most important group of these, which includes the Drummond and Missoula fields, covers nearly all of the southwest corner of the State, all of the coals present being found in the Tertiary lake beds. To the north near the international boundary are found the Valier, Blackfoot, and Glacier Park fields, besides several other small and unimportant areas.

CLASSIFICATION OF MONTANA COALS.

General Statements.

On the map presented with this paper the coals found in Montana are classified into three grades, lignite, sub-bituminous coal, and bituminous coal, while a fourth type present in the Gallatin field is indicated as probably semi-anthracite or semi-bituminous coal, the exact determination being left in doubt because of the lack of information concerning the fuel found there.

The classification of bituminous coals and the coals of lower than bituminous grade, which form by far the greater part of the total coals found in Montana, has always been a very difficult question, for these various types of fuels grade imperceptibly into one another, there being no natural line of separation between them. Chemical analyses alone are not adequate, for the differences between the various grades are physical as well as chemical, so that no scheme of classification based entirely on chemical analyses will be completely successful. As a rule, the lower-grade coals contain much more moisture than those of the bituminous class, but the moisture content of individual samples varies so irregularly and depends so much upon the condition of the mine sampling that it is not an entirely safe criterion

on which to base distinctions, but if the averages of a large number of samples from each type of coal be compared, all the coals being sampled in the same manner, the moisture content of the lower-grade fuels is distinctly higher than that of the higher-grade coals.

An excellent criterion for separating the high-grade coals from those poorer in quality, which can be depended upon more than any other, is the manner of weathering, especially in the fracturing developed in the coal as it gradually dries. In bituminous coals the fractures thus developed will correspond with the cleavage and the resulting fragments will remain prismatic in shape, even to the finest dust particles. On the other hand, the lower-grade coals will check and fracture irregularly, and the resulting fragments will be irregular in outline.

By using the above criteria, a large group of low-grade coals which are very black and frequently of brilliant luster, with fairly well developed cleavage, thus closely resembling true bituminous coals on casual examination, have been separated from the class of bituminous coals, and the term "sub-bituminous" applied to them. About 45 per cent. of the total coal tonnage of Montana consists of this type of coal, which is often called "black lignite" or "lignitic" coal; but the term "lignite" or "lignitic" is not appropriate, for the reason that in no sense are these coals woody, as is implied by the term "lignite." The west one-half of the Eastern Plains region, the Bull Mountain field, Milk River field, and Red Lodge field, all contain this type of coal, while the Great Falls, Lewistown, Bridger, Trail Creek, and Electric fields are the principal areas supplying coal that is truly bituminous in grade. True lignite, which is brown, lusterless, and distinctly woody in texture, occurs only along the eastern border of the State, making up over one-half of the tonnage estimated to be present in the Eastern Plains region.

Comparison of Chemical Analyses and Heating Values.

A comparison of chemical analyses and heating values of coals from 11 of the more important Montana coal fields and of coals from six other fields lying outside of the State, which either come into direct competition with the Montana coals or are accepted standards in the coal trade, is shown in Fig. 3. Proximate analyses giving the ash, fixed carbon, volatile matter, and moisture content of the coals, together with sulphur determinations, are used for a comparison of the chemical character of the various fuels, Table I. The heating value recorded in British thermal units is given in two forms, one for the air-dried sample, which very closely represents the coal as received by the consumer, after drying during shipment and while
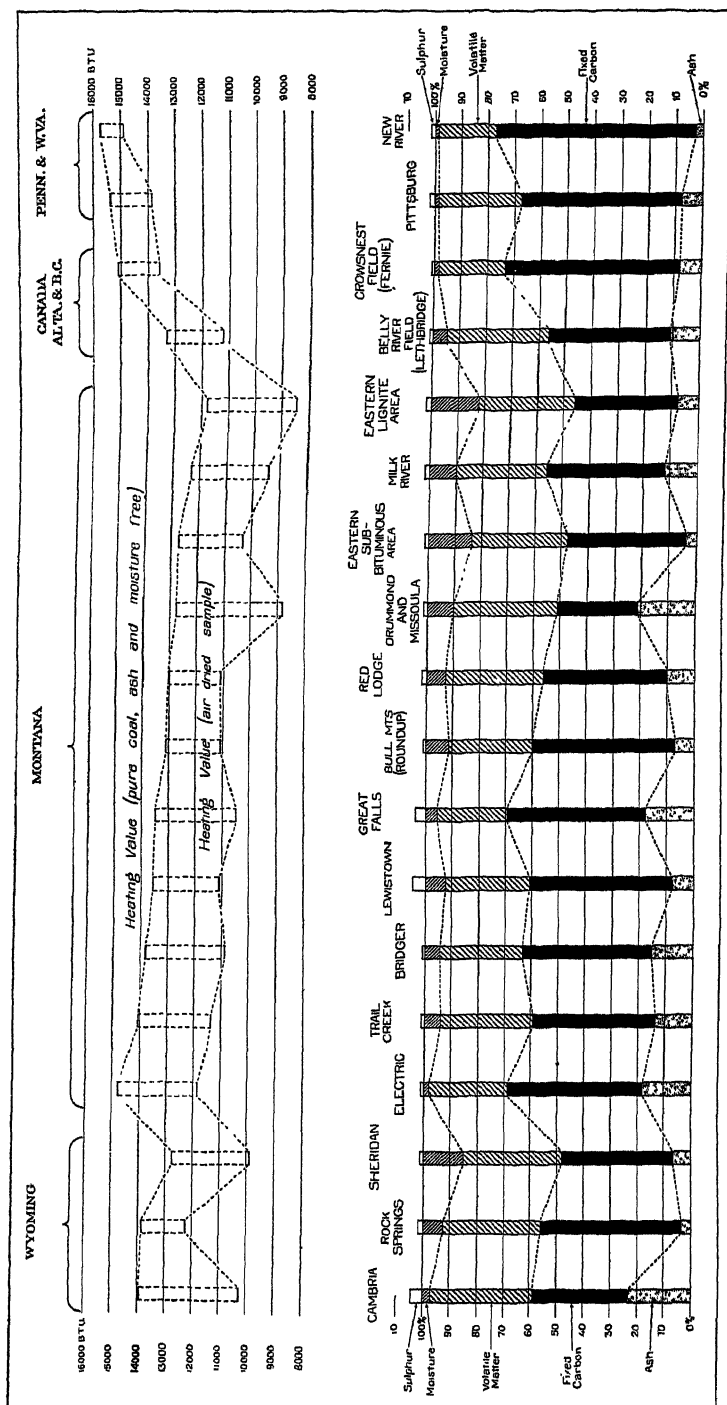
FIG. 3.—HEATING VALUE, PROXIMATE ANALYSES ON AIR-DRIED BASIS, AND SULPHUR CONTENTS OF MONTANA COALS COMPARED WITH COALS FROM ADJACENT COAL-PRODUCING REGIONS IN WYOMING AND CANADA, AND WITH STANDARD EASTERN COALS. (From analyses by the U. S. Geological Survey, U. S. Bureau of Mines, and Department of Mines, Canada.)

The results presented are averages of a number of samples taken from all parts of each field.

being held in stock, and the other for theoretically pure coal from which the ash and moisture content have been removed, obtained by recalculating the results of the determination on an air-dried basis. This second value fairly represents the relative value of the fuel constituents of each of the coals taken alone and is a true measure of the degree of advancement each coal has attained in the process of transformation from original vegetable matter into coal. In every case the results represent the averages obtained from a large number of analyses of coals from all parts of each field and are believed thoroughly to represent the average of the coal that can be produced. All of the analyses and the procedure followed on sampling were made under identical standards, with the exception of the two Canadian coals, which were sampled and analyzed under standards slightly different from those used in the United States, but not different enough to materially affect the results.

TABLE I.—*Heating Values, Proximate Analyses on Air-Dried Basis, and Sulphur Contents of Montana Coals, Compared with Coals from Adjacent Coal-Producing Regions in Wyoming and Canada, and with Standard Eastern Coals.*

(From analyses by the U. S. Geological Survey, U. S. Bureau of Mines, and Department of Mines, Canada.)

State	Field.	Moisture.	Volatile Matter.	Fixed Carbon.	Ash.	Sulphur.	B.t.u.	B t u. (Pure Coal—Ash and Moisture Free.)
Wyoming.	Cambria	3.6	37.9	35.3	23.3	4.5	10,220	13,990
	Rock Springs	8.2	36.0	51.9	3.8	1.1	12,230	13,910
	Sheridan	15.8	37.4	40.7	6.1	0.8	9,980	12,780
Montana	Electric	2.4	29.3	50.3	17.9	0.7	11,800	14,820
	Trail Creek	5.6	34.9	45.4	14.1	0.5	11,390	14,120
	Bridger	5.8	31.1	47.6	15.4	0.6	10,870	13,820
	Lewistown	7.6	30.2	51.9	9.8	3.9	11,040	13,640
	Great Falls	4.1	26.7	50.8	18.4	2.9	10,440	13,580
	Bull Mountain (Roundup)	9.4	31.2	52.8	6.6	0.5	11,070	13,180
	Red Lodge	6.7	36.6	46.6	10.2	2.1	11,060	13,120
	Drummond and Missoula	10.6	38.3	29.9	21.2	1.3	8,810	12,890
	Eastern sub-bituminous area.	16.4	35.9	43.2	4.5	0.6	10,250	12,850
	Milk River	10.1	34.0	43.3	12.6	0.8	9,400	12,350
	Eastern lignite area	19.0	35.6	37.0	8.4	1.1	8.510	11,820
Canada, Alberta and British Columbia.	Belly River (Lethbridge)	9.3	38.0	45.7	12.3	1.0	11,200	13,305
	Crowsnest (Fernie)	1.9	25.1	64.9	9.9	0.5	13,650	15,070
Pennsylvania and West Virginia.	Pittsburg	1.3	31.6	58.7	8.5	1.5	13,920	15,410
	New River	0.8	21.1	74.2	3.9	1.2	14,980	15,710

The arrangement of the coals on the diagram presented in Fig. 3, comparing the coals from the various Montana fields, is determined by the relative heating values of the theoretically pure coal—ash and moisture free—which, as stated above, is believed to be a true measure of the relative degree of coalification attained by the different fuels. Considered on this basis, the coals, as arranged, show a fairly equable gradation in values from 14,820 B.t.u. for the Electric field, which stands first on the list, to a value of 11,820 B.t.u. for the lignite from the Eastern Plains region, which stands last, the most rapid drop in the gradation being from 14,820 B.t.u. for the Electric field to 14,120 for the Trail Creek coals, showing that the coal of the Electric field stands easily above all of the other coals of Montana in the degree of coalification attained. On this basis, going from the highest grade downward, the bituminous coals of the State rank as follows: (1) Electric field, (2) Trail Creek field, (3) Bridger field, (4) Lewistown field, (5) Great Falls field.

Among the coals of the sub-bituminous class, the Bull Mountain coal ranks first, followed in the order named by the coals from the Red Lodge, Drummond, and Missoula, Eastern Plains region, and Milk River fields, which are in turn followed by the lignite from the Eastern Plains region.

On the other hand, if these coals are compared on the basis of the heating values of the air-dried sample, which more nearly represents the coal as used by the consumer, several striking exceptions to the above appear, which in every case seem to be explained by examining the ash contents of the coals in question. The ash simply occupies the place of combustible matter and thus exerts a purely negative value on the fuel. This is probably better illustrated by comparing the averages shown on the diagram obtained by the Cambria and Rock Springs coals from Wyoming than by any coals in Montana. On comparing the heating value of the ash and moisture free coal, the Cambria ranks above the Rock Springs, although its heating value on the air-dried basis is only 10,220 B.t.u., compared with 12,230 for the Rock Springs coal, the difference being due to an ash content of 23.3 per cent., as compared with 3.8 in the Rock Springs coal. The first change to be made in the order of the Montana coals on this basis is that the coal for the Lewistown field, averaging a greater heating value, should be placed above the Bridger coal, because of an ash content of 9.8 per cent. for the former, against 15.4 per cent. for the latter. Again, the Bull Mountain and Red Lodge coals, both of which belong to the sub-bituminous class on the basis of their B.t.u. values for an air-dried sample, rank distinctly above

the Great Falls and Bridger coals, the difference again being due to the varying ash content. The last example is shown by the averages for the coals of the Drummond and Missoula field, which should rank below the sub-bituminous coal, both from the Eastern Plains region and the Milk River field, because of the very high ash percentage, 21.2, found in them.

On studying the moisture content of the Montana coals, it soon becomes evident that the arrangement of the coals inversely according to percentage of moisture in them, would agree in a general way with the arrangement as determined by the heating values of the pure coal—ash and moisture free. The coals of the Electric field show the lowest moisture content, and the lignite of the Eastern Plains region the highest.

On comparing the Montana bituminous coals with those from nearby States, it is seen that both the Electric and Trail Creek coals have heating values for theoretically pure fuel in excess of either the Cambria or Rock Springs coal from Wyoming, but, on the other hand, the low percentage of ash in the Rock Springs coal gives it a much higher heating value on the air-dried basis than any of the bituminous coals in Montana. The sub-bituminous coal from the Bull Mountain and Red Lodge fields easily ranks with or above the coal from the Sheridan field in Wyoming, which is more truly comparable with the sub-bituminous coals from the Montana Eastern Plains region.

On comparing the Montana coals with those from adjacent fields in Canada, it is evident that the coals from the Belly River field (Lethbridge) compare favorably with the Bridger and Lethbridge coals, but are not equal to the Electric and Trail Creek coals. None of the Montana fuels can compare with the high-grade coal found in the Crowsnest field in British Columbia and Alberta, while on the other hand the standard coals from Pennsylvania and West Virginia are seen to excel all other coals whose averages are shown.

DESCRIPTIONS OF THE COAL FIELDS.
Fields Containing Bituminous and Higher-Grade Coals.

Bridger Field,[11, 44.]—The Bridger field is located in Carbon county in the southern part of the State and occupies a long narrow area extending northwest from near the State line in T. 9 S., R. 22 E., across the Clark fork of the Yellowstone and Rock creek for about 35 miles. Considerable development has been accomplished in the northern part of the field, centering around the towns of Bridger, Coalville, and Joliet. It is estimated to contain about 120 sq. miles

of area underlain by coals of workable thickness within minable
depth.

The Eagle sandstone and associated formations, consisting of the
Colorado shale below and the Claggett shale and sandy brackish-
water beds above, outcrop across the entire length of the field. The
geologic structure is comparatively simple, all of the rocks present
having a general southwest dip, the inclination varying from 2° to
10°. Faulting, generally trending nearly at right angles to the strike
of the beds, occurs at several localities, the most prominent example
of which is about 6 miles south of Bridger, near the center of T. 7 S.,
R. 23 E., where a depressed block fault, with its longer axis lying
across the strike and widening in the direction of the dip, offsets the
coal found in the Eagle about three-quarters of a mile to the east.
There is another prominent fault about 4 miles north of Bridger that
offsets the same coal bed for nearly 2 miles. As a rule, these faults
will not interfere seriously with mining operations.

All of the coal found in the Bridger field occurs in the Eagle sand-
stone. The formation contains three coal beds separated by massive
sandstone members about 50 ft. in thickness, only one of the coals
being workable at any one point in the field. The south point of
workable coal occurs in T. 9 S., R. 24 E., the bed containing 2 ft.
6 in. of coal in two benches, separated by a 6-in. bone parting. To
the north, in the Clark Fork valley, in the vicinity of Bridger and
Coalville, the coal being worked varies from 4 to 6 ft. in thickness,
usually occurring in three benches, separated by shale or bone part-
ings. On Rock creek, near Joliet, the bed being worked is thinner,
containing only from 2 ft. to 2 ft. 6 in. of coal, mostly broken into
thin benches. The coals of this field are of lower grade than those
being mined at Electric and Trail creek, but compare favorably with
the Lewistown and Great Falls coals.

Stillwater Field.—A small coal field that has remained undescribed
up to the present time, and here called the Stillwater field, is located
in the valley of the Stillwater river, on the boundary line between
Sweetwater and Carbon counties. The field lies about 35 miles
northwest of Red Lodge, near the base of the rugged mountains
surrounding the Yellowstone National Park. It lies almost wholly
within T. 4 S., R. 16 E., and contains about 10 sq. miles of area
underlain by workable coal occurring at the same horizon as that
mined in the Bridger field.

The Eagle sandstone, the coal-bearing formation in this field, is
about 300 ft. thick, and is exposed on either side of the Stillwater
valley for several miles along an irregular line of outcrop, with dips

varying from east to northeast and north in direction. The coal present occurs in a carbonaceous shale zone at the top of the Eagle, which, in turn, is overlain by sandy beds occupying the stratigraphic position of the Claggett shale. A single bed of coal is present on this horizon, averaging from 4 to 6 ft. in thickness, but it is much broken by shale and bone partings, the thickest single bench measured being a little over 3 ft. in thickness.

The grade of coal in this field is about identical with that of the Bridger coals, being a fair grade of bituminous. The heating value is only a few units less than that of the Bridger coal, and the chemical analyses give practically the same percentage of ash in each of the coals.

Electric Field,[7.]—The area of coal-bearing rocks constituting the Electric field lies on the south boundary line of Montana, in Park county, adjacent to the Yellowstone National Park. The field covers about 20 sq. miles, lying in T. 9 S., Rs. 7 and 8 E., along the Yellowstone river and the minor streams leading into it.

The rocks in which the coals of this field are found occur immediately above a mass of dark marine beds easily recognizable as the Colorado shale, and, therefore, occupy a position approximating that of the Eagle, Claggett, and Judith River formations indicated in Fig. 2. Immediately above the Colorado shale is found a thickness of a little over 300 ft. of sandstone that is soft, generally massive, and contains a few shaly layers, and at the top a coal and carbonaceous shale bed 6 ft. in thickness. This agrees fairly closely with the Eagle sandstone as developed farther east in the Stillwater and Bridger fields. Above this basal sandstone are found a series of alternating sandstones and shales containing two more coal zones, but the stratigraphic equivalents of the Claggett and Judith River formations are not distinguishable.

The structure of the field is complex. Described in a general way, it may be said to represent a fault block narrowing to an apex toward the north and depressed relatively many thousand feet. The block is highly folded and faulted, the faults being of both normal and thrust types, with great irregularity in the angles of dip of the fault planes.

The complexity of the structure is the chief difficulty met with in the mines of this field, for in all the underground workings faults of every type are continually being encountered. On the other hand, the high grade of fuel occurring in this field is almost certainly due to the metamorphism resulting from these structural disturbances. This is especially noticeable in some of the mines where anthracitiza-

tion of the coals is attendant upon sharp folds or occurs in the vicinity of faults.

The coals of this field occur in three beds distributed through about 300 ft. of rocks; the uppermost of these three beds, lying a little more than 200 ft. above the bed next below it, is the only one that has been extensively developed, as it alone is prevailingly coking in character. These coals are easily the highest-grade coals in their class occurring in Montana. Their average heating value on the air-dried basis, 11,800 B.t.u., and their average moisture content, 2.4, are the best values obtained from any true bituminous coals in the State. The coals are coking, practically all of the output being coked and marketed for smelter use at Butte and Anaconda.

Trail Creek Field, [6, 12, 45.]—In all of the earlier publications dealing with the coal areas in the vicinity of Trail creek and Livingston the term "Bozeman coal field" is used. When this name was first adopted Bozeman was the principal town in the region, but at present it seems undesirable to continue the use of this name because the coal-bearing rocks are not exposed near Bozeman but are well developed along continuous outcrops in the general vicinity of Trail creek. The field is not a large one, although, because of its proximity to the main line of the Northern Pacific railroad, it was the first coal area to be extensively developed in Montana. The field contains a total area of about 15 sq. miles underlain by workable coal occupying two straight elongated tracts which meet at Chestnut, making a V-shaped outline on the map. The area is partly in Gallatin and partly in Park county and lies midway between Bozeman and Livingston.

The coal-bearing rocks of this field are very similar to those found in the Electric area. Overlying the marine Colorado shale are found from 750 to 900 ft. of sandstones occupying the stratigraphic position of the Eagle sandstone in other parts of Montana and containing several coal beds of commercial importance. Except for the greater thickness, which in a sandstone of its character is not unusual, the rocks agree closely with the Eagle sandstone occurring farther east in the Stillwater and Bridger fields and can be correlated with that formation with considerable certainty. Overlying this coal group is found a thickness of 5,000 ft. or more of rocks predominantly tufaceous in character, consisting of brown shales and sandstones with intercalated agglomerate largely andesitic throughout, occupying the approximate stratigraphic position of the Claggett, Judith River and Bearpaw formations of other parts of Montana, and being a purely local development in the geologic column of this part of the State.

The structure of the field is complex, both folds and faults being numerous. Along both sides of the valley of Trail creek are found two anticlines bringing up Palæozoic rocks and between them a narrow tract of the coal-bearing Cretaceous formations broken by prominent strike faults.

The coals present occur in several irregular beds that are not productive throughout all of the area mapped. Their variability in character and thickness, together with the complex structures present, makes successful mining in the field very difficult. In grade the coals rank next after those from the Electric field in the class of bituminous coals in Montana, having a slightly lower heating value and considerably greater moisture content than the Electric coals.

Development has continued in this field for many years and it is the only coal area in Montana from which a large part of the easily accessible coal has already been mined.

Gallatin Field, [27,32.]—Coal has been known to be present in the group of mountains lying between the West Gallatin river and the Madison river in the southern part of Gallatin and Madison counties, Montana, for over 30 years, but few or no data have been published concerning it. Two areas are definitely known to be coal-bearing, one on the headwaters of Jackass creek north of Lone mountain, and the other on the headwaters of Dodge creek about 10 miles southeast of the mountain mass known as the Sphinx. The country is rugged and inaccessible, the nearest railroad point being more than 30 miles distant, so that the field has remained undeveloped, in spite of the fact that coals approaching anthracite or semi-anthracite in grade and easily the highest class of fuels found in Montana are known to occur there.

The formation containing the coal in this field is probably identical or very similar to the coal-bearing group found in the Electric field, which lies only about 30 miles to the southeast. Colorado shale is known to be present and above it, as in the Electric field, a considerable thickness of sandy and shale beds containing the coals. In the Dodge Creek area three beds from 4 to 6 ft. thick are known to be present. The main structural feature in each of these coal-bearing areas is a broad open syncline more or less flexed by minor undulations and breaks in the strata. The area on the headwaters of Jackass creek lies adjacent to a large igneous mass of porphyrite many miles in extent and this occurrence probably accounts for the presence of the very high-grade coals reported. A single proximate analysis of coal from the field has been published, [32] giving the following: Moisture, 5.3; volatile matter, 5.6; fixed carbon, 84.7; and

ash, 4.4. This gives a fuel ratio of 15.1, and indicates that the sample analyzed is either anthracite or semi-anthracite coal.

Lombard Field,[41.]—Coal is known to be present probably in commercial quantities in the vicinity of Lombard in Gallatin and Broadwater counties. The area in which the coal occurs lies between Lombard and Toston and occupies about 6 sq. miles. The Kootenai formation is known to be present here and it probably contains the coal-bearing rocks found in the district. The structure is complex, the strata being much broken, so that some of the coal present which is reported to be coking has been altered to such an extent that it is essentially graphite.

Great Falls Field, [14, 15, 22, 23.]—The Great Falls field as generally known at present comprises the large coal-bearing area extending east from the Missouri river in the vicinity of Great Falls for a distance of 60 miles along the base of the Little Belt mountains to a point beyond Stanford. It merges into the area of the Lewistown field, from which it is not divided by any sharp natural boundary. The field lies partly in Cascade and partly in Fergus county.

Throughout this field the coal occurs at one horizon about 60 ft. above the base in the Kootenai (Lower Cretaceous) formation. Coal of workable thickness is not contained, however, at all points on this horizon, but varies locally, there being three principal areas containing minable coal in the field. The first and most important of these areas is in the vicinity of Belt and Sand Coulee and contains 230 sq. miles underlain by workable coal, from which by far the greater part of the production of the field has been mined. The second area, containing 38 sq. miles of workable coal, lies in the vicinity of Spion Kop and Geyser, and the third, with 48 sq. miles of workable coal land, is situated near Stanford. The Kootenai formation as developed in this field has a thickness of about 475 ft. and consists mainly of sandstone and sandy shale occurring in alternate succession. It rests conformably on variegated sandy shales and sandstone of Juriassic age and is overlain by the bluish gray shale of the Colorado formation.

The geologic structure of the field is comparatively simple. The rocks dip at small angles to the northeast away from the adjoining mountains. In a narrow zone bordering the Little Belt range the rocks dip from 10° to 15°, but in passing outward under the plains they flatten rapidly, so that in the coal-bearing areas the dips rarely exceed 4°. However, there are many minor rolls and undulations in the strata, most of which are not perceptible to the casual observer. Minor faults are also more or less common throughout the coal area, especially in the vicinity of Belt, where they have been troublesome in mining operations.

In the vicinity of Sand Coulee the one coal-bearing bed of commercial importance present consists of coal interbedded with layers of bony shale and clay, the coal content ranging from 6 to 14 ft. in thickness in different parts of the district. The arrangement of the coal is in two principal benches, the upper much thicker than the lower. In the vicinity of Belt the coal bed averages about 6 ft. in thickness, occurring in three benches separated by partings of bone. At the Geyser locality the bed averages from 3 to 6 ft., while near Stanford it ranges from 6 to 18 ft., including many partings present.

The coals of the Great Falls field are to be regarded as medium-grade bituminous. They should probably rank the lowest of the true bituminous coals in the State, having lower heating values both on the air-dried basis and on the basis of theoretically pure coal than both the Lewistown and Bridger coals. Their ash content is high, averaging over 18 per cent., and, occurring in conjunction with considerable sulphur in the form of pyrite nodules, renders it necessary to wash these coals before placing them on the market.

Lewistown Field,[3, 4, 12].—The Lewistown coal field as generally known comprises not only the limited district near Lewistown where considerable development has taken place, but also the western extension of that district along the north slopes of the Big Snowy and Little Belt mountains, to where it joins the Great Falls field. It is situated in the center of Montana, including a part of Fergus county and a few square miles in the north part of Meagher county. The greater part of the field lies in the Judith basin, a name applied to the principal drainage area of the Judith river. In all there are about 150 sq. miles of the area in this field that are underlain by workable coals within minable depth.

The rocks encountered in the Lewistown field range in age from Lower Palæozoic to Quaternary, but all of the workable coal in the region occurs at the single horizon found near the base of the Kootenai formation. On the whole, the Kootenai in this field is very similar to that of the Great Falls field except that it is slightly thicker, averaging a little more than 500 ft. The coal horizon, occurring about 90 ft. above the base of the formation, is easily traceable by means of the prominent pebbly and gritty sandstone member about 50 ft. thick, lying immediately above the coal, and generally standing out in bold ridges timbered with pines.

The structure in the western part of the field is relatively simple, the beds present dipping to the north at slight angles away from the Little Belt and Big Snowy mountains. In the eastern part of the field the principal structural features are a group of laccoliths,

the largest of which form the Judith and Moccasin mountains, while the smaller ones are merely small dome-shaped uplifts not over one mile across and often without topographic expression. Erosion has removed the softer Cretaceous rocks from these uplifts, exposing Palæozoic or igneous formations, and as a result the coal-bearing rocks encircle them, the continuity of the coal outcrop being broken here and there by irregular igneous masses.

As in the Great Falls field to the west, the coals in the Lewistown field are not continuous throughout the area, but are productive only in limited districts with unproductive areas between them. The most important of these districts are: (1) the Buffalo Creek district, on the north slope of the Little Belt mountains; (2) the Rock Creek district, on the north slope of the Big Snowy mountains; (3) the Lewistown district, on Big Spring creek just above Lewistown; and (4) the Macdonald Creek district, the largest of all of these, lying on the east edge of the field south of the Judith mountains.

The coals where mined at present vary from 2.5 to 8 ft. in thickness, and, as in the Great Falls field, are usually broken by partings of clay and bone. The coal is a medium-grade bituminous in character and very similar to the Great Falls coal except that it is rather more free from impurities, the ash content averaging almost 10 per cent. lower than the Great Falls coals. It is persistently banded in appearance, with alternating layers of bright and dull coal. The sulphur content is relatively high, averaging over 4 per cent.

Valier Field.—A small heretofore undescribed coal field occupies a small area near the town of Valier in Teton county in the northwest part of the State. The field includes about 6 sq. miles of area that is underlain by workable coal, and although comparatively small is of considerable local importance because of the scarcity of coal in that region,

The coal occurs in a single workable bed at the top of the Eagle sandstone, which is typically developed here, and therefore occupies the same stratigraphic position as the coal mined in the Bridger, Stillwater, and possibly Electric and Trail Creek fields in the southern part of the State. The structure is very simple, the rocks lying so nearly flat and undisturbed that the slight westward inclination of the strata is barely perceptible.

The single coal bed present is thin, rarely being more than 30 in. in thickness, and averaging only about 2 ft. The coal, however, is of good quality, comparing favorably with that mined from the Bridger field and being of slightly better quality than the coal from

the Lewistown field. The ash content is lower and the heating value decidedly higher than that of the coals from the Great Falls field.

Blackfoot Field.—The Blackfoot field is another area that has remained undescribed up to the present time. It lies within 15 miles of the international boundary in the northwest part of Teton county and should develop into considerable importance as a local source of fuel. The field occupies a long narrow area extending in a northwest-southeast direction for about 15 miles across the upper branches of the south fork of the Milk river and is practically undeveloped.

The coal present lies at a single horizon occurring at the top of a sandstone formation locally known as the Horsethief sandstone, lying immediately above the Bearpaw shale, which is the same in stratigraphic position as that occupied by the Lennep sandstone found in the upper part of the Musselshell valley north of the Crazy mountains. The structure of the field is complex, for it lies in a belt of folded and faulted rocks lying at the base of the Rocky mountains. The faults are mostly of the overthrust type. This complexity of structure will make any extensive mining operations very difficult to carry out, but on the other hand the dynamic disturbances have served to alter the coal into a relatively high-grade fuel. The thickest coal found measures 3.5 ft. in thickness, but the average thickness of the workable coal is not over 2.5 ft.

The coal is of excellent quality; the heating value of the theoretically pure coal is 13,890 B.t.u., and considered on this basis it ranks with the better grade bituminous coals of Montana, being slightly above the Bridger coal in grade.

Fields Containing Sub-Bituminous Coal.

Bull Mountain Field, [12, 21, 37, 54.]—The Bull Mountain field is located in the central part of Montana, practically all of it lying in the northern part of Yellowstone county except for a few small areas that lie north of the Musselshell river in Fergus county. Its length from north to south is nearly 25 miles and from east to west 35 to 40 miles. Its total area is about 750 sq. miles, practically all of which is land underlain by workable coal. An earlier statement assigns a total of about 55 sq. miles of coal-bearing area to this field.

Nearly all of the workable coals found occur in the Fort Union formation. It forms the uppermost group of rocks exposed in the field and consists of a thickness of 1,650 ft. of yellowish sandstone and shales interstratified with coal beds. It is prolifically coal-bearing, containing at least 20 beds attaining a thickness of more than 2 ft. of coal. The beds are most numerous in the upper part of the

formation, where they occur at intervals of 50 ft. or less, while in the lower part the intervals are 100 ft. or more.

The geologic structure of the field is simple, the rocks lying practically flat for the most part. The slight dips present form a shallow synclinal basin in the central part of the field, having a general northwest axial trend and a rather accentuated dip at its northwestern extremity. The syncline merges on its northern border into an anticline whose flanks dip about 5°, while this anticline is paralleled on the north by a smaller but sharper syncline. The rocks are practically undisturbed even by minor faults, which with the slight dips present makes the structural conditions ideal for mining.

The coal of this field is a high-grade sub-bituminous very nearly approaching the grade of true bituminous coal. Its rather poor stocking quality is the chief factor that prevents it from being classed as a bituminous coal. It is pitch black to brownish black in color and has a dark brown to black streak. It shows lustrous bands alternating with coal of a dull satiny luster, containing mineral charcoal or " mother " coal.

The average heating value of this fuel on an air-dried basis is greater than that of both the Bridger and Great Falls coals and is about the same as that of the Lewistown and Red Lodge coals, but on the basis of theoretically pure coal the heating value is lower than any of these, except the Red Lodge coal. The Bull Mountain coals with those from Red Lodge should rank first among the sub-bituminous coals found in Montana.

Red Lodge Field, [11, 51, 53].—The Red Lodge field was formerly called the Rocky Ford field, but is now better known by the designation given above from the town of Red Lodge, which was the place of original development and still continues to be the chief mining center. It is situated at the foot of the Beartooth mountains in Carbon county, between the Yellowstone river and the Clark fork, one of the tributaries of the Yellowstone. The field extends for a distance of 8 miles from north to south and an equal distance from east to west. About one-half of this area or 32 sq. miles is underlain by workable coal.

The rocks outcropping in this field belong almost entirely to the Fort Union formation. The sandstones and shales of this formation comprise a mass of rocks 8,500 ft. in thickness in which carbonaceous shale and coal beds are intercalated in various horizons. Workable coals, however, do not occur throughout this immense thickness of rocks, but are confined to a productive member 825 ft. thick, lying between a lower barren member 5,700 ft. thick and an upper barren

member about 2,000 ft. thick. The sandstone and shale of the productive member of the formation closely resemble the rocks from the barren portions and do not seem to indicate any essential difference in the conditions of deposition, the carbonaceous shales and the workable coals present alone serving to distinguish the member. A section of the coal-bearing portion of the rocks present in the field as exposed on the east side of Rock creek near Red Lodge gave a total of 81 ft. of coal distributed in 15 beds in a total thickness of 800 ft. of rocks. This very large amount of coal gives a fair idea of the prolifically coal-bearing character of the Fort Union formation.

A peculiar feature of this field is the presence of a number of dikes of camptonite cutting through the sedimentary rocks and forming low, dark-colored ridges extending in a northwest-southeast direction across the field. They have had little or no metamorphic effect upon the surrounding rocks and where they have cut the coal beds little or no anthracitization has taken place, nothing but charred coal being found.

Structurally, the rocks form a part of an eroded monocline dipping southwestward, which is abruptly terminated at the base of the Beartooth range by a fault having a throw of several thousand feet, bringing Carboniferous limestones in contact with the Fort Union rocks. The southwestward dip of the beds is variable, being 18° at Red Lodge, but gradually flattening going southwestward until they are nearly horizontal near the south limit of the field, there being many minor undulations in the strata in this distance. The Bridger field lying only 12 miles to the northeast occupies the same southwestward dipping monocline present in the Red Lodge field, the only difference being that the coals at Bridger lie at a horizon very much lower than those at Red Lodge.

The coals present were apparently deposited in basins believed to have been shallower toward the southwest because all of the coals thin in that direction and merge into carbonaceous shale, but on the whole the beds present are very persistent and are not to be classed as lenticular deposits. The coals have a black color, pitchy luster and well developed, though irregular, joints. They are medium in hardness and relatively free from injurious impurities. They are about as bright in appearance as some of the bituminous coals of Pennsylvania, and, unlike some of the sub-bituminous coals, do not lose their luster when exposed to the air for a short time. In heating value these coals excel the bituminous coals from the Bridger and Great Falls fields, but their poor stocking qualities and heating value on the basis of theoretically pure coal place them in the sub-bituminous class.

At the present time this field is entering a period of extensive development which promises a continued and steady production for many years.

Missoula and Drummond Fields.—These two fields are isolated areas lying in the mountainous part of western Montana in Missoula and Granite counties. They are the only two areas which up to this time have produced coal in commercial quantities out of the large number of isolated fields containing coal in the Tertiary lake beds in this part of the State.

The coal occurs in the part of the Tertiary lake beds that is approximately White River in age, being found in the lower 100 ft. of the rocks, which are mainly massive tuffs composed in part of clay. The coals are apt to be lenticular, showing considerable variability in thickness. Natural exposures are scarce because the coal crumbles rapidly in weathering. It is distinctly banded, showing alternate layers of dull to bright black coal, and has a conchoidal fracture, occasionally showing a tough woody structure. The coal is distinctly sub-bituminous in grade, and some of it approaches a true lignite.

Milk River Field, [25.]—The Milk River field occupies a large portion of Chouteau county in north-central Montana, and practically all of it lies in the drainage basin of Milk river. By no means all of the large area included in this field is underlain by workable coal, but the districts containing minable fuel are so numerous and so widely scattered over the entire field that it is best treated as a single large coal field.

The formations found in the Milk River field are almost identical with those found farther south in central Montana along the Missoula and Musselshell River valleys, which may be spoken of as the standard section of the Cretaceous rocks for Montana. The Eagle sandstone, Claggett formation, Judith River formation, and Bearpaw shale are present and present little variation from their development to the south.

All of the coals of workable thickness in this field occur within 150 ft. of the top of the Judith River formation. They are lenticular in shape and show variable thicknesses up to a total of 9 ft. of coal on a single outcrop. The beds are noticeably thinner and of poorer grade in the eastern part of the field. Generally, there, is only one bed at a single locality of workable thickness, but in a few places as many as four beds are found.

All of these coals may be classed as a fair grade of sub-bituminous. They are pitch black to brownish black in color and have a brown streak, their luster being bright and waxy. Chemical analyses show

that they are comparatively low in sulphur and high in moisture and ash. The heating value of these coals, taken on the basis of a theoretically pure coal, averages only 12,350 B.t.u., which is the lowest value obtained for any of the sub-bituminous coals in Montana, although on the basis of the air-dried sample the heating value exceeds that of the coal from the Missoula and Drummond fields.

The Eastern Plains Region of Lignite and Sub-Bituminous Coal,

1, 2, 5, 16, 17, 40.

This very large coal and lignite bearing area covers nearly all of the east one-half of Montana, including practically all of Valley, Dawson, Custer and Rosebud counties within its boundaries. It is estimated to contain 365 of the total 380 billion short tons of lignite and coal found in Montana. About 40 per cent. of this total tonnage is estimated to consist of sub-bituminous coal, while the remainder is a true lignite. Its total tonnage and area greatly exceed that of all the other fields in the State taken together, but up to the present time it has furnished almost nothing to the annual production of coal in the State.

All but a very small proportion of the coal in this region occurs in the Fort Union formation, which here has its typical development. The number and distribution of the coals present are about equal or slightly exceed the number of beds and the amount of coal found in the Bull Mountain field, which can properly be regarded as an outlier of the great eastern area which has been isolated by erosion since the beds were first deposited. The structure over all of the area is very simple, the rocks for the most part lying practically horizontal or else flexed into very broad gentle undulations that are barely perceptible on casual examination. The only exception to this is an anticline whose axis, trending in a northwest-southeast direction almost exactly paralleled by that of the Black Hills and Bighorn uplifts, passes through a point a few miles southwest of Glendive in Dawson county and extends for over 60 miles to the southeast, to the North Dakota boundary. This fold is asymmetric, the steeper-dipping limb lying on the southwest side, where the beds are inclined at angles varying from 10° to 30° but flatten very rapidly in traveling away from the fold toward the southwest.

The fuel in the eastern one-half of this region is a true lignite and constitutes the lowest grade of fuel found in Montana. Its heating value for the theoretically pure coal, ash and moisture free, averages 11,820 B.t.u., while on the air-dried basis the result is 8,505 B.t.u. Both of these figures are the lowest obtained for any of the coaly fuels found in Montana. In general, the lignite is brown, lusterless

and woody, showing the grain of the wood and frequently the out-
lines of entire tree trunks from which all or a large part of the ma-
terial was formed. Going toward the west across this region the
lignite changes gradually into a black, shiny, sub-bituminous coal.
The change is so gradual that probably no two observers would agree
as to the place where the line of division should be drawn. As the
brown color disappears the lignite loses more and more of its woody
character. Near Glendive it is black but lusterless, at Miles City its
color is a little more pronounced, and much of it is black and shiny,
while on the extreme western edge of the area it has practically lost
all traces of its woody structure and has obtained a brilliant luster
throughout, being a true sub-bituminous coal.

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A Note on the Occurrence and Manufacture of Refractories in Montana.

BY W. H. GUNNISS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

WHEN the copper smelters were built, in Montana, all of the refractory products which were used in their construction were shipped in from Eastern factories. It was apparent that if a material suitable for the manufacture of these products could be found within the State, a saving of freight could be effected and a new industry built up.

Prospecting for such materials was started and experiments were made with samples from numerous deposits. Considerable effort and funds were expended in an attempt to manufacture silica brick from Dillon quartz, a light colored sandstone that is found near Dillon, Mont., on the Oregon Shortline railroad. As a brick material the Dillon quartz was a failure, but it has proved very satisfactory as a silica cement. It is now being used in laying silica brick and for reverberatory furnace bottoms.

A bed of quartzite that is particularly suited to the manufacture of high-grade silica brick was finally discovered at Browns Spur, 6 miles west of Anaconda. The material from this deposit is now being used in the manufacture of the Anaconda silica brick, which have gained a wide reputation for heat-resisting and lasting qualities. The analysis of an average sample of this quartz is: SiO_2, 96.7; Fe_2O_3, 0.3; Al_2O_3, 2.5; CaO, 0.3 per cent. The very low percentage of fluxing ingredients makes this a most refractory material.

All of the Anaconda silica brick are hand molded. Various types of brick machines have been tried, but the hand-mold process proved the best suited to the manufacture of silica brick from this material. The silica is first put through a jaw crusher and is then dumped into wet pans, where 4 per cent. of lime paste is added, to produce a bond. Without the lime the material would not be plastic enough to mold. After a few minutes' grinding in the wet pans the material is conveyed to the molder, who forces it into the molds by hand. The brick are then dumped on pallets and placed in the drier, where they are left until dry enough to be set in the kilns. Some of the brick are re-pressed before being dried. They are burned about five days, in round down-draft kilns. It is not necessary to have as much heat in the kilns as the brick will be subjected to when in use in the furnaces. Montana refractories are used in smelters throughout the West.

Prospecting for fire clays was also carried on. After experimenting with clays from several different deposits, it was found that a mixture of

40 per cent. of a flint fire clay (which is quarried on Lost creek, 4 miles east of Anaconda) and 60 per cent. of a plastic fire clay (which is mined near Armington, Mont.) made an excellent fire brick.

The flint clay from Lost creek is unlike the typical flint clays of Pennsylvania, though a very good fire clay. It is a light colored rock, analyzing: SiO_2, 71.6; Fe_2O_3, 1.3; Al_2O_3, 20.2; CaO, 0.4 per cent.

The plastic fire clay mined at Armington is of a very good quality. It is found in the Kootenai formation. The deposit has a thickness of from 4 to 5 ft., and occurs about 26 ft. above the Kootenai coal horizon. It is slate colored, fine grained and homogeneous, breaking with a sub-conchoidal fracture. The analysis of this clay is: SiO_2, 55.7; Al_2O_3, 30.8; Fe_2O_3, 1.1; CaO, 0.2; MgO, 0.5; alkalies, 2.1; loss on ignition, 10.2 per cent. The fire brick are manufactured at the Anaconda plant by the same process as is used in making the silica brick. At the Great Falls plant of the Anaconda Copper Mining Co., the fire brick are made by what is known as the dry press process.

Comparative tests have been made by placing fire brick from Eastern factories beside the Montana fire brick, in the furnaces. The result has been that the Montana brick stood the tests as well as any of the celebrated refractory brick manufactured in the East.

Refractory wares are manufactured in Butte from a clay quarried on Lost creek near the deposit worked by the Anaconda Copper Mining Co. A sample of this clay gives the following analysis: SiO_2, 72.8; Fe_2O_3, 3; Al_2O_3, 23.7 per cent.

Building brick are manufactured in all the largest cities of the State, the red surface clays being used in most cases, though shales suitable for the manufacture of paving brick are found in some localities and used for making building materials.

That the clay resources of the State have not been further developed is due largely to the fact that many of the deposits are at present inaccessible, but as the railroads branch out the conditions for exploitation become more favorable and these clays will be of commercial value.

DISCUSSION.

PROF. J. W. RICHARDS, South Bethlehem, Pa. :—In regard to the physical nature of quartz for making silica brick, is the coarse-grained preferable ?

E. P. MATHEWSON, Anaconda, Mont. :—A dense quartz is preferable. The quartzite is the actual material used by the Anaconda Company. The sandstone referred to, the Dillon quartz, was a very pure sandstone and had hardly any impurities in it, and the brick made from that particular material, while apparently beautiful while coming out of the kiln, and true, immediately on heating in the furnace puffed up half an inch and were cracked open.

Cement Materials and the Manufacture of Portland Cement in Montana.

BY W. H. ANDREWS,* TRIDENT, MONT.

(Butte Meeting, August, 1913.)

THE constantly increasing consumption of Portland cement in the State makes the above subject of particular interest at this time. The increasing demand is due to the rapid settling of the country and the fact that new uses are being found frequently for the above named commercial article. The industry is one that is bound to grow as the population increases, and this being the case it may be well at this time to consider briefly the matter of the chemical and physical composition of proper raw materials.

A cement mixture is one that contains approximately 75 per cent. carbonate of lime, 15 per cent. silica, and 5 to 7 per cent. alumina and iron oxides; the other elements being magnesium carbonate and small quantities of the rarer elements generally associated with limestone deposits. It can be readily seen from the above composition that it is possible to build up this mixture from a number of different materials.

First.—A rock high in carbonate of lime can be mixed with clay or shale in such proportions as to bring about the desired composition; the chief difficulty in the building up of such a mixture is that it is hard to find a clay or shale in which the silica-alumina ratio lies between 4 to 1 and 3 to 1 and at the same time shows physical characteristics which will make certain its adaptability to the practical manufacture of Portland cement. Very often clays or shales are found which are right chemically but contain large quantities of grit or free silica. Such clays or shale must be ground to extreme fineness in the mixture in order to make the silica and alumina combine with the lime when the mixture is burned. On the other hand, clays or shales are often found that have the proper physical characteristics but in which the amount of alumina is too great.

Second.—A mixture can be built up of different limestones some of which run above the 75 per cent. requirement and others below;

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* Non-member.

these limestones, however, must contain the right proportions of silica and alumina and must possess the proper physical characteristics.

Third.—A large percentage of the Portland cement marketed in the country was formerly manufactured from marl and clay, but on account of the low specific gravity of the marl and the large organic content, this method has given way to the making of cement from different kinds of rock.

Fourth.—A limestone containing approximately the 75 per cent. carbonate of lime required is sometimes, but rarely, found. Such a stone is called "cement rock." If such a deposit contains the silica and alumina in the proper ratio and the corrective materials occur in the same quarry, the property is of value. It can be readily seen that a natural mixture can be handled in such a manner as to produce the best possible product with the least trouble in the manufacture.

An example of cement rock is the Trenton rock of the Lehigh valley, Pennsylvania, from which large quantities of Portland cement are manufactured. It must be borne in mind, however, that in all of the above-named methods the materials used must be low in magnesia and free from sulphur.

In regard to the process of manufacture it might be said that the advancement has been remarkable during the past few years. The tendency has been toward fewer units, with larger capacity per unit. One of the large rotary kilns of to-day will do the work of five of the kilns in common use 10 or 12 years ago. The same is true of machinery for the preparation of the raw materials and the grinding of the finished product.

Montana, up to date, has but one cement plant in operation: that of the Three Forks Portland Cement Co., located at Trident, just below the Three Forks of the Missouri river on the Northern Pacific Railway. This plant manufactures the "Red Devil" brand of cement and has been in active operation since June, 1910. This plant uses for its raw materials a cement rock of Devonian age.

The mill was constructed by F. L. Smith & Co. and is an example of an up-to-date plant. The rock is loaded in the quarry by two 80-ton Bucyrus steam shovels into bottom-dump steel cars. These cars are hauled by a steam locomotive a short distance to the crusher house, where they are discharged into a No. 9 crusher which has a much greater capacity than the mill's regular requirement. This crusher reduces the rock to about 2-in. size and under; the oversize is sent to a smaller crusher before going to the blending bins.

After being crushed, the rock is elevated to the top of what are known as blending bins—these are eight in number, having a capacity of 200 tons each, four being on one side of the scale hopper and four on the other side. The rock running higher in carbonate of lime than the required mix is kept on one side of the mixing scales and the rock lower in carbonate of lime than the required mix is kept on the other side. As the rock drops to the bins, it passes through equalizing drums and is automatically sampled. The resulting sample is ground and removed to the laboratory for chemical analysis.

The rock in the different bins having been carefully analyzed, the mixture is calculated by the chemist. The rock is then conveyed to the scale hoppers by belt conveyors which travel underneath the bins. Here, the high and low rock is weighed off in the proper proportion. By the method of sampling in use, it is possible at all times to keep the cement mixture the same. From the scale hoppers the rock is elevated to a bin over the drier. The rock in passing through the drier is deprived of its moisture. This puts it in condition for fine grinding. From the rotary drier, which is fired by producer gas, the rock is elevated to bins over grinding mills, called "kominuters."

The kominuters are large drums, revolving on horizontal shafts. The drums are lined with perforated steel plates and surrounded by screens set at such an angle as to return automatically all coarse material to the mill for regrinding. The grinding is accomplished by large steel balls which are given a tumbling action as the mill revolves. The raw rock is ground in these mills to a fineness of about 80 per cent. passing a 20-mesh screen. Samples of the product of these mills are taken every two hours to the laboratory for fineness test and chemical analysis.

From the kominuters the partially ground material is elevated to bins over the tube mills, into which mills it passes for the finished grinding. The tube mills are 22 ft. long by 6 ft. in diameter. These mills are divided into two compartments by a perforated plate placed 5 ft. from the discharge end. The large end of the mill is filled with pebbles and the small compartment with what are known as cylpebs. These cylpebs are small cylindrical pieces of steel. Both compartments are filled nearly half full with pebbles and cylpebs. The mills make about 25 rev. per min., the material being fed constantly through a small screw conveyor from the bin to the pebble end of the mill. The cylpebs increase the efficiency of the mill, making possible larger capacity and greater fineness.

The raw mix, which is finished to a fineness of 90 per cent. passing a 100-mesh sieve, is sampled and analyzed every two hours, during

the 24, and passes through screw conveyors and bucket elevators to the bins over the rotary kilns. The kilns are two in number, 9.5 ft. in diameter by 140 ft. in length and having a combined capacity of 1,600 to 1,800 barrels daily.

The fuel used is powdered coal, mined at Red Lodge. This coal is ground to a fineness of 95 per cent. passing 100-mesh sieve and is blown into the front end of the kilns by an air pressure furnished by large fans. The coal therefore burns with a flame very much like gas, raising the heat in the lower part of the kilns to about 3,000° F.

The raw material is fed into the upper end of the kiln through a screw conveyor and as the kiln is set on a slight angle and revolves slowly, the mix travels toward the discharge end. As it passes through the heat zone the carbonic acid is driven off and the lime carbonate is changed to lime silicates and aluminates. It can be seen that careful preparation of the raw materials is necessary to bring about the proper combination of the lime, silica and alumina. During this process, the powdered mix undergoes a physical as well as a chemical change. It enters the kiln a light-colored powder and is discharged in the form of white-hot clinkers, changing to dark green and black when cooled, and varying in size from that of a small pea to several inches in diameter. Chemically, these clinkers are now Portland cement, but it is necessary to grind them to a very fine powder in order to have commercial Portland cement.

The clinker passing from the kilns, white hot, is caught in a large elevator and carried to a rotary cooler. The cooling is accomplished by natural draft. From the cooler, it is elevated and discharged into the storage yard. From the yard, the clinker is taken through a tunnel by belt conveyor and elevators to the final grinding room. The grinding of the finished product is accomplished by kominuters and tube mills, which have been described in the grinding of the raw materials. It is necessary at this stage of the manufacture to add 2 to 3 per cent. of raw gypsum to regulate the setting time.

As the finished cement drops from the tube mills it is sampled every two hours and taken to the laboratory for physical tests and chemical analysis. The cement then passes by means of screw and belt conveyors and bucket elevators to the stock house, where it is stored until required for shipment.

The stock house is of reinforced concrete and has a capacity of 40,000 barrels of cement. It is divided into five bins or compartments. From these stock-house bins, the cement is conveyed and elevated to eight packing hoppers, each having a capacity of about 1,000 barrels, from which it is fed to the packers.

The product is shipped in duck and burlap bags, 95 lb. to the sack, and four sacks to the barrel. As each car is loaded, samples representing the contents are taken and sent to the laboratory for confirming tests. From the time the raw material leaves the quarry until the finished product goes into the cars, it is not touched by hand.

Cement machinery is ponderous and the wear and tear tremendous —competent superintendence and skilled operators are necessary in the manufacturing end. As has been mentioned elsewhere in this article, the process of cement manufacture has undergone a wonderful evolution during the past few years. Old style mills cannot keep pace with the more modern ones, either in the quantity of output or quality of the finished product. Specifications are now drawn up with a view to finding every possible weakness in the cement as marketed. The strength required is greater, and realizing the importance of sand-carrying power, the railroads and different municipalities are demanding greater fineness. The Auto Clave test has been invented to prove beyond question the absolute soundness of Portland cement. These requirements can only be met by the mills that are modern or have kept up-to-date and the public is guaranteed safety when it uses cement that has passed standard specifications, provided the work is properly done and the proper aggregates are used.

Cement materials are rather widely scattered throughout the State, the difficulty being to find all the materials together and the location suitable with regard to market. Doubtless, in time other plants will be built in the State, but at present the one mill in operation is easily able to supply the State and in fact is obliged to market part of its product in adjoining States.

The Metaline Plant of the Inland Portland Cement Co., Metaline Falls, Wash.

BY MILO W. KREJCI, GREAT FALLS, MONT.

THE plant and quarries of the 'Inland Portland Cement Co. are located at Metaline Falls, Wash., about 128 miles north of Spokane, on the Pend Oreille river, and within 10 miles of the Canadian border. The plant is one of a chain of cement plants controlled by the Lehigh Portland Cement Co., of Allentown, Pa. Work on this plant was commenced in June, 1910, and was completed in April, 1911, but on account of some delays in the construction of the power plant, the works were not put into operation until August, 1911. Since then it has been in continuous operation and has furnished the Inland Empire with a high grade of cement. Fig. 1 is a view and Fig. 2 is a plan of the works.

Water Power Development.—The source of the water supply for the hydro-electric plant of the Inland Portland Cement Co. is Sullivan lake, a body of water approximately 5.5 miles long and 0.5 mile wide, situated in the Kaniksu Forest Reserve. A wooden crib dam was installed so as to raise the elevation of the lake 25 ft., and the additional storage capacity was taken up by the diversion of Sullivan creek into the lake by means of a wooden flume, 0.25 mile long, 6 by 8 ft. in cross-section, and a ditch 1,200 ft. long.

The water from the lake follows the natural water course to an intermediate storage reservoir, which was created by the construction of an earthen dam. This pond was used in connection with the saw mill which was installed at this point.

Besides the earth dam, there was also built a crib, at the point shown in the sketch map, Fig. 3. This crib dam is 35 ft. high. From this point the excess water follows the natural course of Sullivan creek to the Pend Oreille river.

The water is taken from this intermediage storage by means of a 6 by 8 ft. wooden flume approximately 13,000 ft. long. This flume is built up on the side hill, piling being driven most of the way on
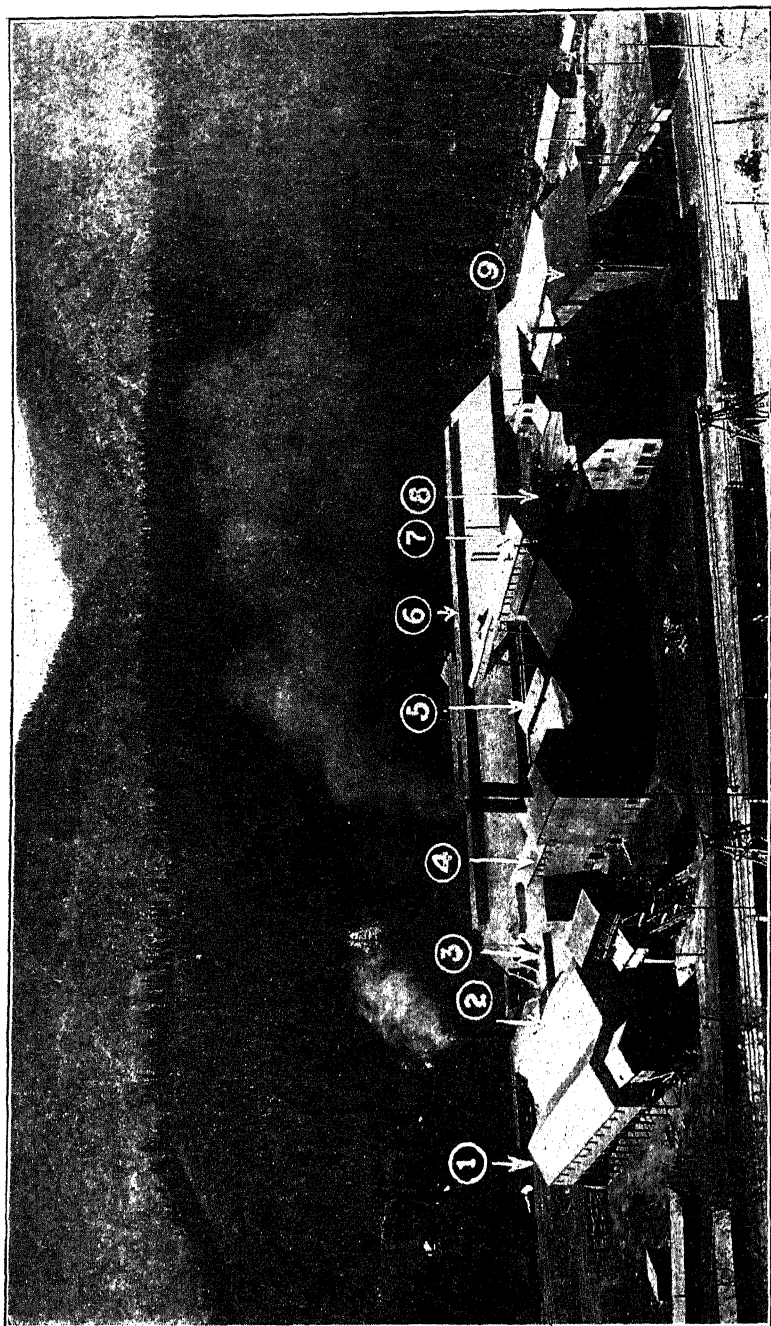
1, Lime rock.   2, Shale.   3, Mixing of shale and lime rock.   4, Grinding of mixture.   5, Cement kilns.   6, Cement storage and sacking.   7, Final grinding of clinker.   8, Clinker coolers.   9, Coal grinding and drying.

FIG. 1.—GENERAL VIEW OF THE WORKS OF THE INLAND PORTLAND CEMENT CO., METALINE FALLS, WASH.

account of the lack of rock foundation. From the end of the flume line is a ditch 850 ft. long, which leads into the head gates. The water at this point has a vertical drop of 41 ft. and an angle of 45° in a tunnel through the hill, down to the elevation of the wheels. The tunnel is lined with 36-in. steel pipe throughout its entire length. The total head is 465 ft. and the working pressure at the wheels is 204 lb. per square inch.

Power House.—In the power house are installed two 1,865 kv-a. Westinghouse generators, connected directly to Pelton water wheels of the double-bucket type, two nozzles to each wheel, one 3.75 in. in
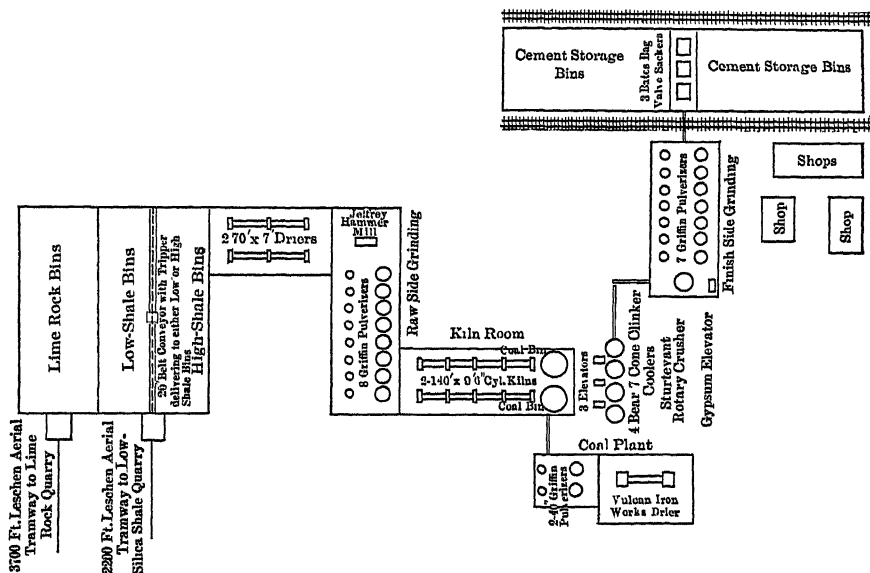
FIG. 2.—PLAN OF PLANT OF INLAND PORTLAND CEMENT CO.

diameter and the other a needle-valve nozzle. The generators are driven at a speed of 360 rev. per min. and generate 260 volts. In addition to the two generators, there are two 125-kw. exciters.

The current for quarry purposes is transmitted at 2,300 volts, 500 kw. being transformed to 550 volts at the quarry. The transmission from the power house to the cement plant is at 2,300 volts and is transformed to 550 volts with two water-cooled transformers of 1,000 kw. capacity. There is also an oil transformer for the lighting circuits.

Quarries.—The company has three quarries of raw materials from which the cement is made :

1. A limestone quarry, about 3,700 ft. east of the plant.

2. A low-silica shale quarry, about 2,200 ft. east of the plant.

3. A high-silica shale quarry, about 4 miles south of the plant.

The raw materials are quarried by means of drilling with air drills of the Ingersoll-Rand and Jeffrey types and blasted down with dynamite.

1. The limestone quarry is a very extensive one and is operated in the open-quarry fashion, the rock being loaded into hopper-bottom
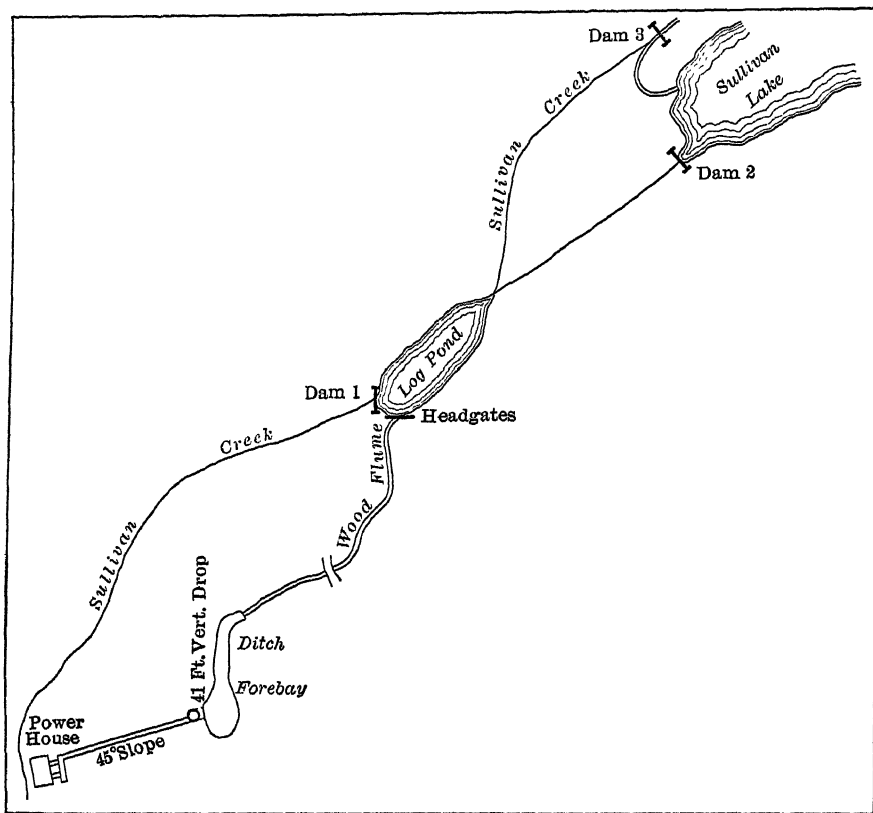
FIG. 3.—SKETCH-MAP SHOWING WATER POWER DEVELOPMENT OF THE INLAND PORTLAND CEMENT CO.

cars by means of a Marion-50 steam shovel and hauled to the crushing plant by horses.

The crushing plant is but a short distance from the quarry. It is equipped with a No. 9 Gates gyratory crusher, whose product is delivered to two No. 5 Gates gyratory crushers, where it broken down to about 1.25-in. size. The crushers are all independently driven and have a capacity of 200 tons per hour.

The broken lime rock is delivered to the loading bins, whence it is transferred to the cement plant by means of a Leschen aerial tramway, 3,700 ft. long. The tramway is equipped with 27 buckets on the line, each bucket having a capacity of 1,200 lb. These buckets are automatically dumped into the lime rock storage at the cement plant. This tramway requires no power to operate it, being operated by gravity, and in fact it develops 16 h-p. It is controlled with a triple brake.

2. The low-silica (calcareous) shale quarry is located about 2,200 ft. east of the cement works and about 300 ft. from the power house. This shale contains from 30 to 55 per cent. of silica and comes from strata about 400 ft. below the limestone.

Open-cut quarrying is employed, the rock being loaded by hand into tram cars and taken to a loading bin, from which it is delivered into buckets on a Leschen aerial tramway, which conveys it to a No. 5 Gates gyratory crusher at the shale storage trestle.

The shale tramway is driven by a 30-h-p. motor. No particular spacing for the buckets is required. About 12 h-p. is actually required to operate this tram.

The product of the crusher falls on a conveyor belt fitted with a tripper, which dumps it on the low-silica shale storage pile.

3. The high-silica shale is obtained from an open cut along the Idaho & Washington railroad, about 4 miles south of the cement works, and is loaded into hopper-bottom cars by means of a Marion-28 steam shovel.

The shale is broken down in a No. 5 Gates crusher, and is delivered to the storage pile by means of a 20-in. belt conveyor. It contains from 55 to 75 per cent. of silica.

Drying Limestone and Shale.—The high-silica and low-silica shales are now drawn from chutes, in tunnels underneath the storage piles, on to 20-in. conveyors and delivered, in the proper volumetric proportions required in the mix, to an elevator, which feeds into a bin preceding the driers.

In a like manner, the limestone is fed on to a 20-in. conveyor belt and elevated to a bin preceding the drier used for drying the crushed limestone.

The Driers.—There are two cylindrical driers, one on limestone and one on shale, each being 70 ft. long and 7 ft. in diameter, set on an inclination of ⅜ in. to the foot; Z bars are riveted longitudinally on the inside, which turn the material into the path of the hot gases. They were built by the Vulcan Iron Works.

The dried materials are now elevated to separate bins and fed

therefrom to an automatic weighing machine, in the proper quantities required in the "mix." The automatic weighing machine was furnished by the Automatic Weighing Machine Co., of Newark, N. J.

Pulverizing the Mix.—From the weighing machine the mixture is conveyed on a 20-in. belt conveyor to a type B Jeffrey hammer mill, which reduces it to about 0.25 in. size. The mill is run at about 1,200 rev. per min. and is operated by a 100-h-p. motor.

The product of the hammer mill is elevated and then conveyed to bins which feed eight 40-in. Giant Griffin pulverizers, driven by 75-h-p. vertical type alternating current motors. The average power consumption is about 68 h-p. per mill on about 100 tons of mix per day.

. The resulting product, now ground so that 96 per cent. will go through a 100-mesh sieve, discharges from the mills into a screw conveyor which carries it to the kiln building.

The Kilns.—The pulverized material from the Griffin mills is elevated to the kiln bins, which feed into two rotary kilns, 140 ft. long and 9 ft. 6 in. in diameter, set at an inclination of ⅜ in. per foot and lined with silica brick, the feed being automatic. The kilns are each driven by a 50-h-p. variable-speed motor, which regulates the speed of calcination of the mix, under the supervision of a skilled operator.

Coal Drying and Pulverizing.—The drying and grinding of the coal used in the firing of the kilns is performed in a building adjacent to the kilns. The fuel used is Roundup coal from Montana and is an ideal coal for cement work, on account of its low ash, high volatile matter (35 to 40 per cent.) and high calorific value. When this coal is not available, Canadian coals are used. The coal cost is from $4.85 to $5 per ton.

The coal is unloaded from railroad cars into a bin under the railroad tracks, and conveyed on an inclined 20-in. conveyor belt to the coal storage pile. From here the coal is drawn in a tunnel, which runs underneath the storage pile, from chutes to a 20-in. belt conveyor, which delivers it to an elevator. It is discharged from the elevator into a bin which feeds into a Vulcan Iron Works type of drier, 30 ft. long and 3 ft. 6 in. in diameter.

The dried coal is elevated to two steel bins, which feed into two 40-in. Giant Griffin pulverizers, which reduce the coal so that 95 per cent. will pass through a 100-mesh sieve, after which it is elevated and then conveyed in a screw conveyor to two steel bins, immediately in front of the kilns, from which it is automatically drawn and blown into the kilns with air furnished by two Sturtevant blowers.

Operation of the Kilns.—The air, coal, and speed of the kilns are

under absolute control of the operator at all times. The air compressed by the blowers is drawn from the clinker pit and in consequence is slightly warmed.

The speed of the kilns varies; usually it is one revolution in from 1 to 2 min.

The feed pipes for fuel to the kilns are to one side of the center, placed so that the flame does not continuously impinge on the superheated "mix," and the mixture does not fall into the flame as the kiln revolves, sintering the same to the point of incipient fusion. The temperature of the kilns is from 2,700° to 3,000° F.

The brick used in the fuel-feed end of the kilns, for about 50 ft., is a special high-grade fire brick very low in iron, in order to withstand the high heat.

The kilns each have an actual daily capacity of about 800 barrels, and are operated continuously with the exception of a semi-annual shut down, for the purpose of repairs, in July and December.

The clinker is discharged into the clinker pit, a narrow brick chamber, below the discharge end of the kilns.

Cooling the Clinker.—The clinker from the pit is discharged into three chain bucket elevators, which feed into four Bear cooling towers, consisting of seven cones each. These towers are shown in Fig. 4. Water is fed into the buckets of the elevators, as they leave the pit, which slakes any free lime present, and partly cools and disintegrates the clinker. In its passage through the cooling towers, through which is distributed 40,000 cu. ft. of air per minute, the clinker is cooled down and finally discharges into a steel pan conveyor.

In case there is a break down beyond this point, or too much clinker is being made, the clinker is delivered to the clinker storage pile, through steel chutes, where it remains until it can be put through the finishing process.

Addition of Gypsum and Final Grinding.—The gypsum, which is added to the cement in order to give it the desired set, is put in with the clinker immediately following the coolers.

The clinker which has been cooled, either in the cooling towers or at the storage pile, is delivered to an automatic revolving scale. This machine automatically weighs 500 lb. of clinker and then dumps. Each time 500 lb. is weighed the required amount of gypsum is added, varying from 1.75 to 2 per cent.; roughly, a shovelful to each 500 lb. The gypsum used is furnished by the U. S. Gypsum Co., of Great Falls, Mont. It comes in a ground form in railroad cars, which unload into a bin underneath the tracks, and is delivered to a storage bin, from which it is drawn as required.

The clinker with the gypsum added is now sent to a Sturtevant rotary crusher by way of a 20-in. belt conveyor. The product of the preliminary crusher is elevated to the top of the building and conveyed in a screw conveyor to bins over the final grinding machines.

The final grinding is performed in seven 40-in. Giant Griffin mills, driven by 75-h-p. vertical type alternating current motors. The power consumption is from 72 to 78 h-p. per mill. The screens on the mills are 40 mesh, the resulting product (cement) being 85 per cent. through 200-mesh and 95 per cent. through 100-mesh screens.
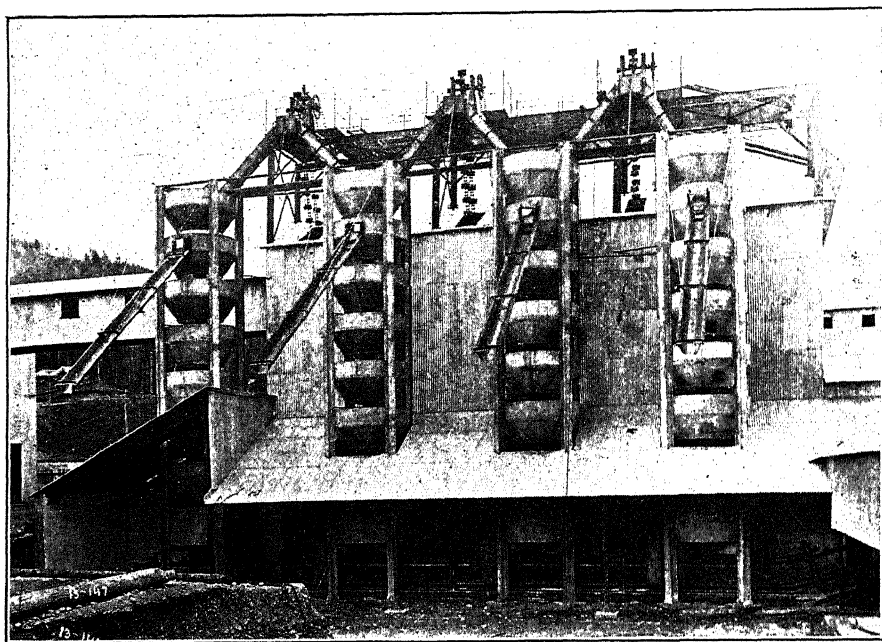
FIG. 4.—BEAR CLINKER COOLING TOWERS.

The finished cement is now sent to the cement storage house by means of an elevator and conveyors. The cement storage is divided into two parts, each capable of holding 110,000 barrels of cement. This gives an ample capacity for holding the cement until it has been thoroughly analyzed and tested.

Sacking the Cement.—The cement after having been thoroughly tested is drawn from the bottom of the storage bins, through chutes, to screw conveyors which operate in tunnels, and is delivered to one of two elevators, which discharge into bins over three sacking machines, which were furnished by the Bates Bag Valve Co., of

Limestone Quarry
1 Marion Steam Shovel
Tram Cars
1 No.9 Gates Gyratory Crusher
2 No 5 Gates Gyratory Crushers
20 Conveyor
Loading Bins
1 3700' Leschen Aerial Tramway
1 Large Receiving Bin or Storage Pile
20" Conveyor
Elevator
Bin
1 70' x 7' Vulcan Rotary Drier
Elevator
Bin

High Silica Shale Quarry
55 to 65% Si O₂
1 Marion Steam Shovel
R.R. Dump Cars
Receiving Chute
Elevator
1 No.5 Gates Gyratory Crusher
20" Conveyor Belt
1 Large Receiver Bin
20" Conveyor

Low Silica Shale Quarry
45 to 55% Si O₂
Tram Cars
Loading Bins
1 2200' Leschen Aerial Tramway
1 Large Receiving Bin or Storage Pile
20" Conveyor
Elevator
Bin
1 70' x 7' Vulcan Rotary Drier
Elevator
Bins

Automatic Weighing Machine
20" Belt Conveyor
1 Jeffrey Hammer Mill
Elevator
20" Belt Conveyor
Bins
8 40" Griffin Pulverizers
Screw Conveyor
Elevator
Steel Bin
Automatic Feeder
2-140' x 9'6' Rotary Kilns
1 Clinker Pit
3 Steel Chain Conveyors
Water added in Elevator Cups
4 Bear 7-Cone Clinker Cooling Towers
Steel Pan Conveyor
Cross Conveyor

Coal Flow Sheet
R.R. Dump Cars
Bins under Trestle
20" Belt Conveyor
Coal Storage Pile
Via chute in tunnel to
20" Belt Conveyor
Elevator
1 Vulcan Coal Drier
Elevator
Steel Bins
2 40" Griffin Mills
Elevator
Steel Screw Conveyor
2 Steel Bins
Blown through pipe with air
from Sturtevant Fans
Coal feed regulated by speed change

Gypsum Flow Sheet
R R Cars
Bin under Trestle
Elevator
Storage Bins
Chute
1 Shovelful for each 500 lb. Clinker
Or 1.75 Per Cent
Excess to Clinker Storage
Via chute in tunnel to
20 Belt Conveyor

Automatic Revolving Weighing Machine
Weighs 500 lbs. and discharges
20" Belt Conveyor
1 Sturtevant Rotary Crusher
Elevator
20" Belt Conveyor
Clinker Bins
7 Griffin Pulverizers
Screw Conveyors
Elevator
20" Belt Conveyor
Cement Storage Bins
Via chute in tunnel to
Screw Conveyor
2 Elevators
Steel Bins
3 Bates Bag Valve Sackers
20' Conveyors to R.R. Cars

Sack Flow Sheet
New Sacks   Old Sacks Returned
                   Rotary Carpet Cleaner
                   Sorted
Good   Sacks to be   Worthless
Sacks   Mended   Sacks
2 Sack Tying Stands-25 Sacks
Hand Truck

FIG. 5.—FLOW SHEET OF THE PLANT OF THE INLAND PORTLAND CEMENT CO.

Chicago. From the sackers the sacked cement is discharged on a conveyor and delivered directly into railroad cars.

The equipment includes a carpet-cleaning type revolving drum for cleaning returned sacks, a sack repair room, and stands for tying sacks before sending them to the Bates sackers.

The output of the plant is from 1,500 to 1,600 barrels per day. The mill is operated in two shifts of 12 hr., labor being obtained at from $2.50 to $2.75 per day of 10 hr. A complete flow sheet of the operation of the plant is given in Fig. 5.

Analyses of Raw Materials and Cement.—The following analyses represent samples of the lime rock, high-silica and low-silica shale, and the finished cement:

	Lime-stone.	Calcareous Shale.			High-Silica Shale.		Inland Cement.
		No. 1.	No. 2.	No. 3.	No. 1.	No. 2.	
Loss on ignition (CO_2)..	38.98	19.77	21.01	15.2	3.73	5.96	....
Silica (SiO_2)...............	7.00	32.48	31.96	40.5	74.40	71.40	22.48
Iron (Fe_2O_3)................ }	5.32	19.92	18.00	5.2	} 13.00	14.04	{ 2.2
Alumina (Al_2O_3)......... }				16.4			6.4
Lime (CaO)................	47.50	20.41	22.61	15.4	1.57	3.07	62.27
Magnesia (MgO).........	1.52	3.42	2.97	2.7	2.28	3.26	3.54
Sulphur trioxide (SO_3)..	....	....	....	....	....	....	1.50
Cementation index......	....	....	....	....	....	....	1.08

General.—Since commencing operations the Inland Portland Cement Co. has supplied all the cement for the city of Spokane and vicinity, or the Inland Empire, as it is called.

Practically all of the recent power installations in that vicinity have used this cement. The most notable of all is the Long Lake power dam of the Washington Power Co., in which about 275,000 barrels of cement will be used.

The equipment of the Inland plant includes a well-equipped testing and chemical laboratory, shops, etc.

I wish to express my thanks to the officials of the Inland Portland Cement Co., for their kind aid in furnishing data for this paper, particularly Dan R. Brown, Treasurer, and Irving J. Kohler, Superintendent.

Lead-Silver Mines of Gilmore, Lemhi County, Idaho.

BY RALPH NICHOLS, GILMORE, IDAHO.

THE mines are near the town of Gilmore, in the Texas mining district. This district was organized in 1880. The present producing mines are near the terminus of the Gilmore & Pittsburg railroad. This railroad connects with the main line of the Oregon Short Line at Armstead, Mont. The distance from Gilmore to Armstead is 70 miles.

The town of Gilmore is near the head of the Lemhi river at an elevation of 7,000 ft., while the mines are from 400 to 1,300 ft. higher. The range of mountains upon which the mines are situated forms the water shed for the Little Lost river on the west and the Lemhi river on the east. The winters are severe and the snow fall is heavy, but the mines ship right through the year. There is an abundance of timber suitable for all mining purposes in the immediate vicinity. Water is piped into the camp from a lake about 2.5 miles distant. This lake is about 1,400 ft. above the town of Gilmore. The water is exceptionally good for boiler and domestic purposes.

The mining companies now producing are the Pittsburg Idaho Co., Ltd.; the Latest Out Mining & Smelting Co., Ltd.; and the Gilmore Mining Co., Ltd. Besides the above properties, there are a number of prospects in different stages of development.

The producing mines of the district are fissures in limestone, with a general northeast-southwest course and a dip to the west of from 40° to 70°. These fissures are at about right angles to the bedding of the limestones. The veins are regular and the ore shoots strong, both laterally and in depth, so far as developed. The ore shoots have a rake or pitch to the south. The character of the formation is shown in Fig. 1, a section of the Latest Out mine.

Character of Ore.

The ores of the district are oxidized, carrying an excess of iron over the silica (insoluble matter). The vein filling is a mixture of limonite and hematite, which is replaced by shoots of lead-silver ores.

The lead minerals consist of carbonate of lead principally, but with small quantities of sulphide, sulphate, and other lead minerals.

One peculiar feature noticeable in these properties is that the sulphides have gradually decreased with depth. For example, the ore from the upper workings down to 200 ft. in depth would be represented by the following:

Lead. Per Cent.	Silica. Per Cent.	Iron. Per Cent.	Zinc. Per Cent.	Silver. Oz. per Ton.	Gold. Oz per Ton.	Sulphur. Per Cent.
35	20	10	7	16	0.025	0.8

The ore below the 200-ft. level and down to 500 ft. would be represented by the following:

Lead. Per Cent.	Silica. Per Cent.	Iron. Per Cent.	Zinc. Per Cent.	Silver. Oz. per Ton.	Gold. Oz. per Ton.	Sulphur. Per Cent.
32	10	25	3	12	0.025	0.03

FIG. 1.—SECTION OF LATEST OUT MINE.

From the above comparison it will be seen that there is a slight falling off in lead and silver contents with depth. It will be noticed too that there is a smaller quantity of zinc, silica, and sulphur present, with a decided increase in iron content, making the ore a most desirable smelting ore, and it naturally commands a good price in the Salt Lake markets, where such ores are scarce.   The ore is mined in bulk, requiring no concentration or assorting.

Ore Occurrence.

As before stated, the fissures are very regular, but the filling of the vein is a replacement of the limestone on either side of the fissure and sometimes reaches widths of clean ore of from 30 to 40 ft.   The vein filling is a mixture of limonite and hematite replaced by shoots of lead minerals.   The change from lead to iron content is generally very abrupt, so that it is not difficult to break the ore very clean.

The ore is generally shipped in barge lots of from 50 to 500 tons (from one to ten railroad cars) and the lead content varies from 25 to 45 per cent.

History.

The first discoveries in the Texas mining district were made in the year 1880, and at that time little work was done, as the mines were then 80 miles distant from the nearest railroad shipping point (Caucas).

Later, however, in 1886, the Viola Mining & Smelting Co., of Nicholia, Idaho, erected its smelting plant and bought the ores from the Texas district, which gave quite an impetus to prospecting in this section.   This smelting plant was successfully operated for about four years, when the ores of the Viola mine were exhausted and the mines of the district shut down on their work and did little more than the annual assessment work.

In the year 1908 the building of the Gilmore & Pittsburg railroad was assured and at that time the work of developing the leading mines of this section was renewed.

Shipments were resumed in 1910, and from that time to the present the three properties mentioned have collectively shipped more than 100,000 tons of ore.   Of the above amount, the Pittsburg Idaho Co., Ltd., has shipped about 75,000 tons; the Latest Out Mining & Smelting Co., Ltd., has shipped about 27,000 tons; and the Gilmore Mining Co., Ltd., has shipped about 1,200 tons of ore carrying gold values averaging about $12 per ton and an excess of iron over silica of 40 per cent.

The lode being worked upon the Gilmore property contains only a very small quantity of lead and in this respect is quite unlike the other lodes in this district.

When one considers the short time that these properties have been worked and the small amount of development work done upon them the yield has been most satisfactory.

The Tooele Plant of the International Smelting & Refining Co.

BY H. N. THOMSON AND L. T. SICKA, TOOELE, UTAH.

GENERAL.

THE Tooele plant of the International Smelting & Refining Co. is situated at the mouth of Pine canyon, Tooele county, Utah. It is connected with the main line of the San Pedro, Los Angeles & Salt Lake railroad by the Tooele Valley railway, and is about 7 miles from Tooele Junction. Fig. 1 is a general view of the plant.
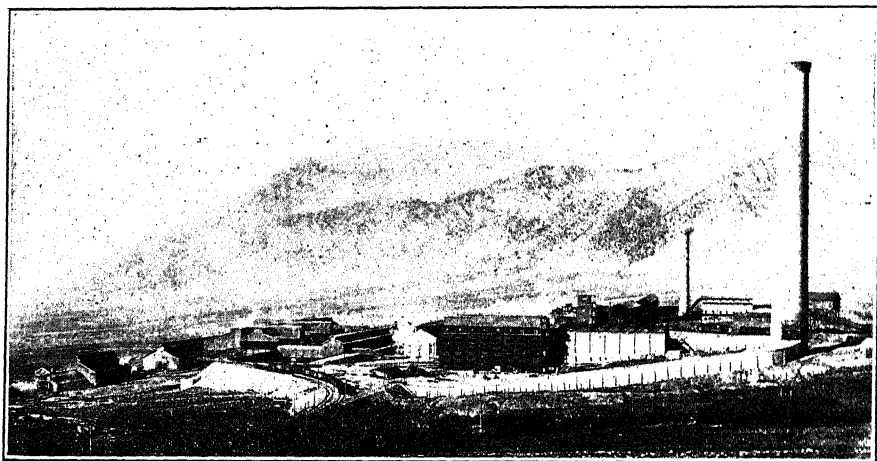
FIG. 1.—TOOELE PLANT OF THE INTERNATIONAL SMELTING & REFINING CO.

The copper plant has been in operation since July, 1910, and the first blast furnace at the lead plant was blown in on Feb. 29, 1912.

The location of the plant was selected on sloping ground with the idea of using gravity to the fullest extent and thus keeping labor at a minimum. Practically the entire plant is constructed of steel, reinforced concrete, and brick.

Bingham ores, both lead and copper, are received over the Utah Consolidated Aerial Tramway. The tram buckets, each containing

about 1,100 lb. of ore, are dumped into wooden terminal bins of 800 tons capacity. Ore is trammed from the terminal to receiving bins in a 50-ton electrically propelled steel bottom-dump car. All other ores, coal, and fluxes are received in railway cars, and delivered directly to the sample mill receiving bins, or to the lead plant charge bins, when material is already of suitable size for shovel sampling and direct sintering or smelting.

Coke is received in railway cars and unloaded from an overhead trestle into a storage pit arranged with concrete tunnel underneath. This tunnel contains a belt conveyor, equipped with a series of receiving hoppers and adjustable feed chutes, by means of which a uniform stream of coke is maintained on the conveyor. This conveyor discharges into a 750-ton circular bin, from which coke for blast-furnace charge is drawn direct. This is equipped with weighing hoppers similar to those under the main bin system.

Receiving Bins.

The capacity of the sample mill receiving bins is about 4,000 tons of ore. These are divided into compartments for the purpose of classification. A portion of the same series of bins is devoted to coal service, and has a capacity of 2,500 tons. The ore is delivered to the sample mill by belt conveyors equipped with automatic feeders. The coal is delivered to the reverberatory furnaces and boiler plant by hopper-bottom tram cars.

Sample Mill and Crushing Plant.

This is situated adjacent to the receiving bins, and consists of two complete independent sections for crushing and sampling. The method of sampling is the standard Brunton system, consisting of four Brunton oscillating time sampling machines in each section. The final sample weighs 3.2 lb. for each ton of ore sampled. The crushing sections are arranged to supply either the copper or the lead plant, and are equipped with necessary crushers, rolls, elevators, trommels, and belt conveyors, to deliver ore of suitable size as required by the different plants.

Adjacent to the mill is a complete pulping and sampling room equipped with necessary fine-grinding machinery and electrical drier. A room is provided for mine representatives from which sampling operations can be watched.

Resample and Converter Ore Bins.

These consist of a series of side-discharge bins, having a capacity of 3,000 tons, divided into compartments for classification and stor-

age of small lots pending acceptance of first sampling. The contents of the bins are drawn into railway or local tram cars and delivered direct to the converter plant, or switched back to the sample mill receiving bins for resample.

Roaster Bins.

These consist of a series of side-discharge bins with a total capacity of 5,500 tons of crushed ore. Each side of the bins discharges on to a belt conveyor by automatic feeder, thence by successive series of belt conveyors to the roaster charge hoppers. The distribution of material to these hoppers is by mechanically propelled discharge trippers.

COPPER PLANT.

Roaster Plant.

This consists of 32 16-ft. MacDougall type roasters, arranged in four rows. Gases from each two rows combine in one elevated brick flue with steel hopper bottom arranged with dust pipes. These flues discharge the gases directly into a dust chamber, 140 by 120 ft., and 30 ft. high above the hopper level. The dust is withdrawn from the chamber into tram cars, through three tunnels arranged with hoppers and discharge gates for this purpose.

Reverberatory Plant.

This plant consists of one 19 by 90 ft. and four 19 by 102 ft. Anaconda type coal-fired reverberatory furnaces. Each furnace is equipped with a 700 h.-p. Stirling waste-heat boiler. One boiler is arranged with underfeed stoker and conveyor system for fuel supply, thus permitting power generation in case of furnace repairs or when any other reason necessitates shutting down the smelting furnace.

Slag is skimmed into 22-ton pots arranged with electrical tipping apparatus, Fig. 2. Ashes are discharged into 7-ton automatic side-tipping cars. Matte is conveyed by clay-lined launders, slope $\frac{1}{8}$ in. per foot, to the ladle, which is poured direct into the converters by crane.

Converting and Casting Plant.

This consists of five stands of horizontal cylindrical type shells, 96 by 150 in. These are electrically operated by individual motors. The converters are all lined with magnesite brick. Air pressure of from 10 to 11 lb. is found to be most satisfactory. The converters are served by a 60-ton electric traveling crane. Siliceous ore is handled in 2-ton capacity boats. Converter slag is cast in beds, loaded

into cars with crane, and shipped to the crushing plant to be sampled and prepared for blast-furnace use. Copper is cast into stationary steel molds, being poured by a 30-ton crane. The converter gases are delivered to the intake of a No. 20 Sirocco fan of 180,000 cu. ft. minimum capacity. This discharges directly into a brick flue leading to the bag house chambers. In case of sudden excessive temperature the gases may be by-passed and discharged directly to the stack.
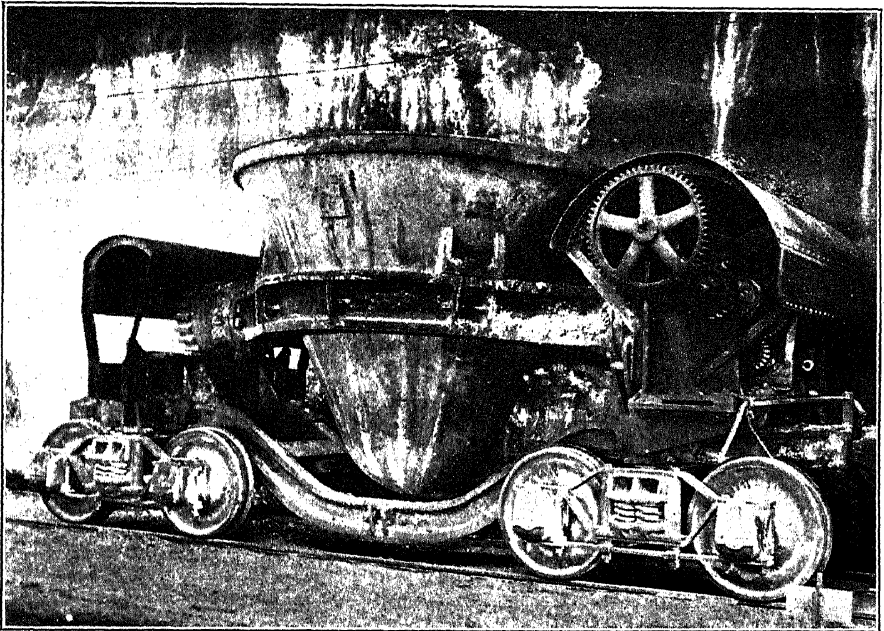
FIG. 2.—22-TON ELECTRICALLY TIPPED SLAG POT.

Converter Bag House.

This building is divided into eight lower chambers, or cellars, covered with a steel thimble floor. The building above the thimble floor consists of one large chamber containing 960 woolen fabric bags, 18 in. in diameter by 30 ft. high. The flow of the gases to the lower chambers is controlled by large externally operated disk valves situated in the main flue. Shaking the bags is accomplished by reverse current system. This comprises a small auxiliary flue situated directly over the main flue, and arranged with openings leading directly into the lower chambers, and controlled with disk valves. One end of this auxiliary flue is connected with the intake of a No. $4\frac{1}{2}$ Sirocco fan, the discharge from which passes directly into the main flue. The bag-shaking system is shown in Fig. 3.

Fume Chamber · Fume Chamber

Bags

Discharge from No. 20 Sirocco Fan

PLAN

1 Set of Valves

Bags

Steel Thimble Floor

Reverse Current Gas Flue

Auxiliary Valve

Open · Closed

No. 4½ Sirocco Suction Fan

Reverse Current Gas Flue

Operation:
For bags in service, main valve open, auxiliary valve closed.
For collapsing bags, main valve closed, auxiliary valve open.

Main Valve

Fume or Burning Chamber

Main Gas Flue

Closed · Open

1 Set of Valves

Main Flue

1 Set of Valves

CROSS-SECTIONAL ELEVATION

SECTIONAL ELEVATION

FIG. 3.—REVERSE CURRENT BAG-SHAKING SYSTEM FOR BAG HOUSES.

Main Building Walls

Handle

2½" Pipe Support

Chain Travel

Chain for Shaking

Steel Spring

Cen. Line of Building

Bag→

2½" Pipe

No. 8 Single Iron Jack Chain

No. 8 Soft Iron Wire

½" Pipe-4½" long

18"
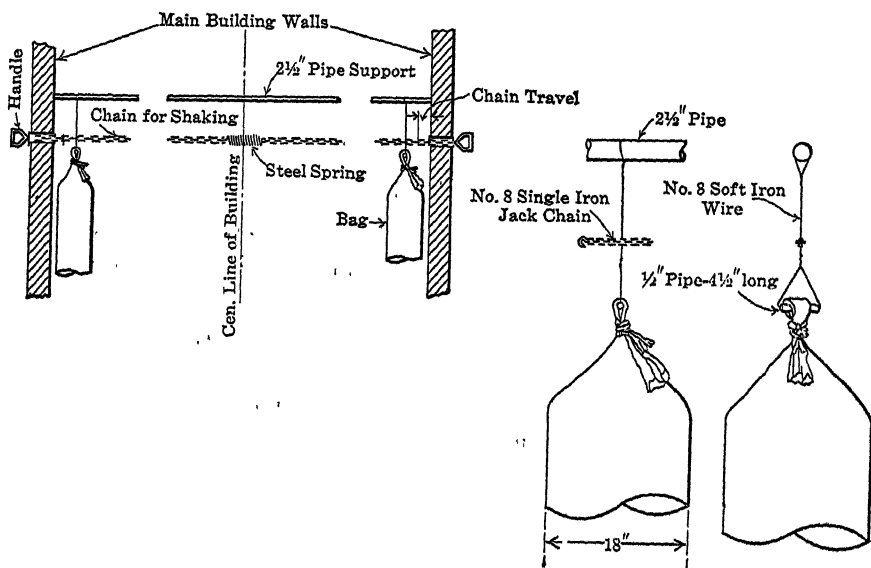
FIG. 4.—AUXILIARY BAG-SHAKING DEVICE FOR BAG HOUSES.

The bags can also be shaken by hand in case of necessity. This is accomplished by means of a small chain fastened to the upper bag connection, and extending through the side walls of the building. A handle is provided at each end. A long spiral steel spring is located centrally on each of the above chains which gives a vibratory shaking. The device is shown in Fig. 4.
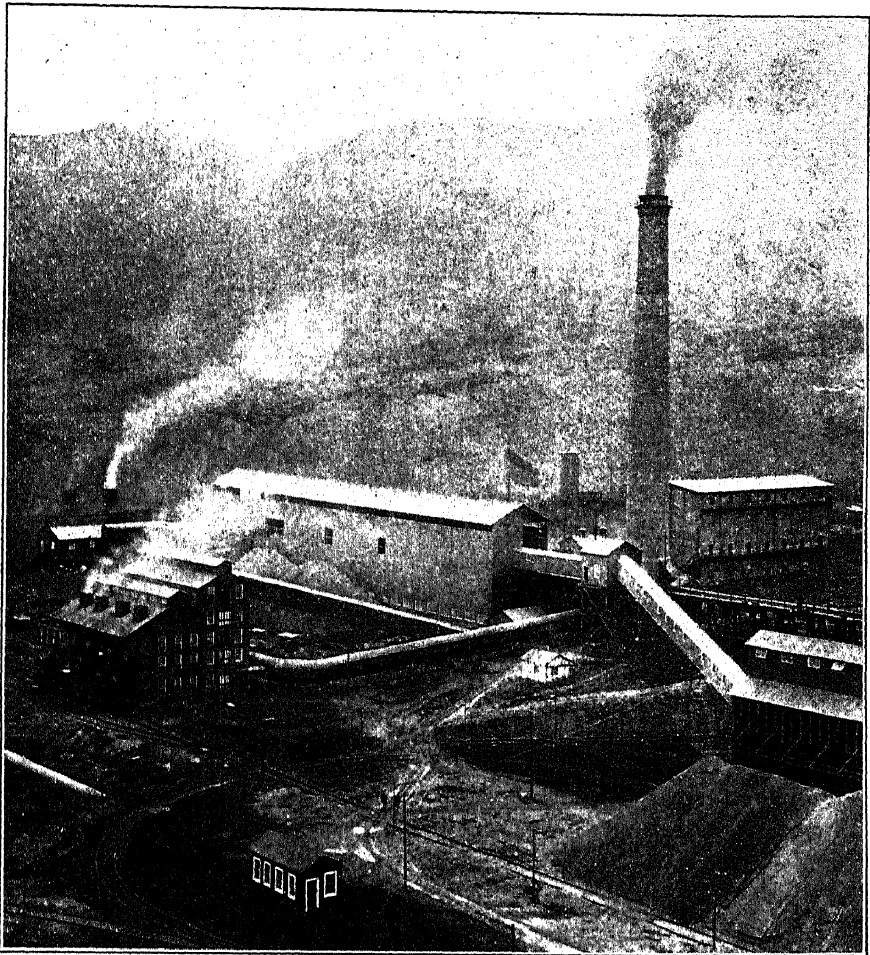
FIG. 5.—GENERAL VIEW OF LEAD PLANT.

Dust accumulation in the lower chambers is removed by a system of screw and belt conveyors discharging directly into railway cars. Gases from the bag house pass through a brick downtake into a brick stack, 15 by 150 ft.

Stack and Flues.

Gases from the reverberatory plant pass to the main copper plant stack by a 20 by 18 ft. brick flue, 1,360 ft. long, with flat arched roof, and gases from the roaster dust chamber through a flue 16 by 16 ft. by 225 ft. long.

The stack is of wire-cut brick construction, 25 ft. in diameter inside, and 350 ft. high. The stack and the flues are shown in Fig. 1. ·

Lead Plant.

Sinter-Plant and Blast-Furnace Charge Bins.

These bins are arranged in two adjacent rows; are 260 ft. long, 32 ft. wide, and divided into 52 compartments. Each compartment is 10 ft. long and 16 ft. wide, with a maximum depth of 24 ft. The bins are hopper bottomed and arranged with independent chutes and circular type hand-operated gates. Each compartment has a capacity of 250 tons of lead ore. The adjacent compartments converge to central discharge chutes.

Both series of compartments are supplied directly by railway cars or overhead conveyors with automatic traveling trippers. Thus, ore or concentrates already fine enough for treatment can be unloaded direct from railroad cars, while all other material comes direct from the crushing plant by a system of 20-in. belt conveyors, 820 ft. long. The entire structure is inclosed. Each pair of compartments is provided with a suspended weighing hopper in conjunction with special Fairbanks suspended-type scales having independent beams, shown in Fig. 6. This arrangement permits material from each compartment to be weighed separately. All scale hoppers have circular type hand-operated gates by which the hopper contents are discharged directly into sinter or blast-furnace charge cars. One end of the installation is reserved for material for sinter-plant charge, while the other is reserved for blast-furnace charge. This makes an extremely flexible system, as material in all 52 compartments is available on short notice. All material being available for either sinter or blast-furnace charge car permits of immediate charge alterations and eliminates possible trouble caused by irregularities in bedded ore. All material for the lead plant can be classified in the 52 compartments, and the material is uniformly spread in the charge cars by running the car slowly forward and back while ore or coke is being discharged from the weighing hopper.

The charge cars are hopper-bottom type with two drop doors. They have pivoted four-wheel trucks at each end, arranged with railway

type motor, geared direct to axle. Each car is thus equipped with two motors, which are operated through a street railway type controller at one end of the car. The drop doors and brake mechanism can be operated from either end of the car. The sinter-plant charge-car body is 12 ft. long and 4 ft. wide, and holds from 3 to 5 tons, depending on the material. The blast-furnace charge-car body is 14 ft. long and 5 ft. wide, and holds from 5 to 8 tons.
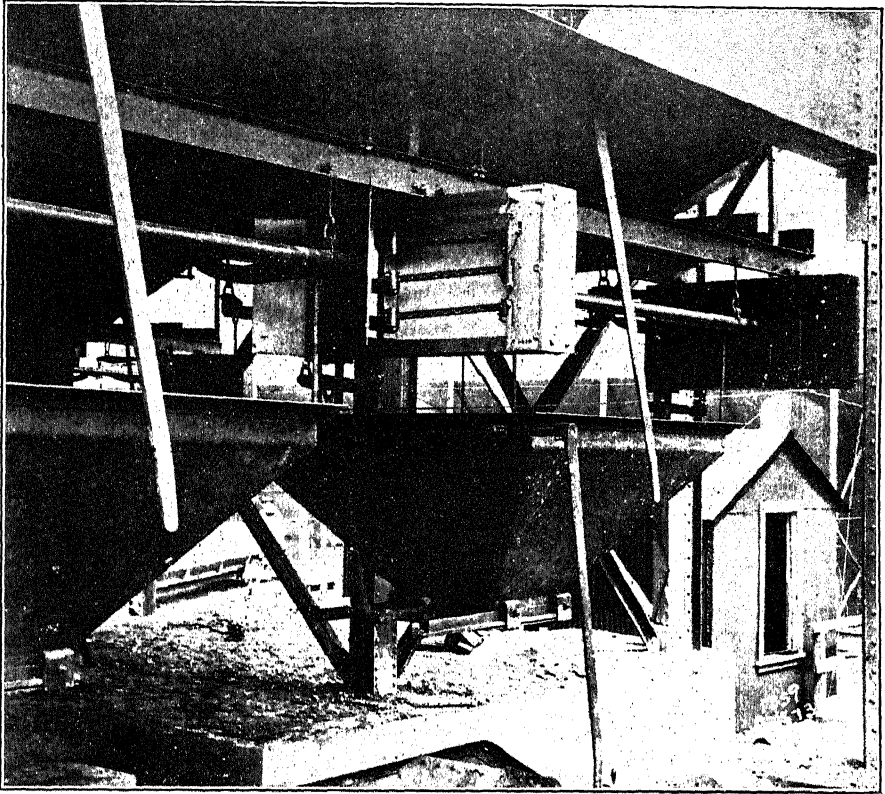
FIG. 6.—SUSPENDED WEIGHING HOPPER AND SCALES.

Sintering Plant.

This is housed in a building 55 ft. wide, 160 ft. long, and 42 ft. high, arranged with ground floor, two main working floors, and upper distributing floor. The plant consists of 10 standard 42 by 264 in. Dwight-Lloyd machines.

The weighed sinter charge is dropped from the car into a steel hopper located at the extreme end of the main charge bins. This hopper is divided into two sections, each fitted with a small belt feeder dis-

charging to one central chute, thence to the main inclined 18-in. belt conveyor. This delivers to a system of cross conveyors aranged with tripper stations located directly opposite each machine feeder hopper.

Each machine is arranged with a special adjustable feed and distributing apparatus. The feed hopper consists of a cylinder with conical bottom holding from 4 to 5 tons of charge. The discharge nozzle is fitted with an adjustable sleeve for regulating the flow of the material on to a horizontal rotating feeder table 3 ft. in diameter. The material is discharged from this table by an adjustable plow and falls directly into a segment of an adjustable inverted cone. This directs the feed to the apex of a raised conical formation in an inclined adjustable chute. The material by this device is distributed uniformly across the entire width of the sintering machine, and falls into a leveling hopper, which is directly over the traveling grates of the sintering machine, as shown in Fig. 7.

The rotating feed table is operated by bevel gears and sprocket chain drive from the sinter-machine drive gear. The charge is ignited by oil burners inclosed in a steel housing. Oil is supplied from four 100-gal. service tanks, which in turn are intermittently supplied by gravity from a 14,000-gal. cylindrical storage tank. This is located in an underground concrete building.

The product of the machines discharges into inclined chutes arranged with movable gates, thence into railway cars. Each machine is driven by an individual 3-h.-p. variable speed back-geared motor, which is controlled on the main operating floor. Each machine has two 10-ft. wind boxes. The suction draft system of the plant consists of five special fans of modified Sirocco type, single intake, with impeller 5 ft. 6 in. in diameter, mounted on a shaft and direct coupled to a 75-h.-p. motor running at a speed of 850 rev. per minute. The capacity is from 8,000 to 16,000 cu. ft. per minute. The draft varies from 5 to 12 in., depending on the porosity of the material on the machine grates. Each fan is connected to the wind boxes of two machines by a 30-in. steel pipe. The gases are discharged from the fan into a modified form of balloon-type steel flue having a continuous hopper bottom fitted with small dust-discharging gates. This flue is 440 ft. long, and leads directly to the chimney.

Blast-Furnace Plant.

The blast-furnace building is 66 ft. wide, 138 ft. long, and 44 ft. high. The plant consists of four furnaces, with a fifth under construction. Two of the furnaces are 45 by 180 in. at the tuyeres, and two are 60 by 180 in. All furnaces are the same height. From the
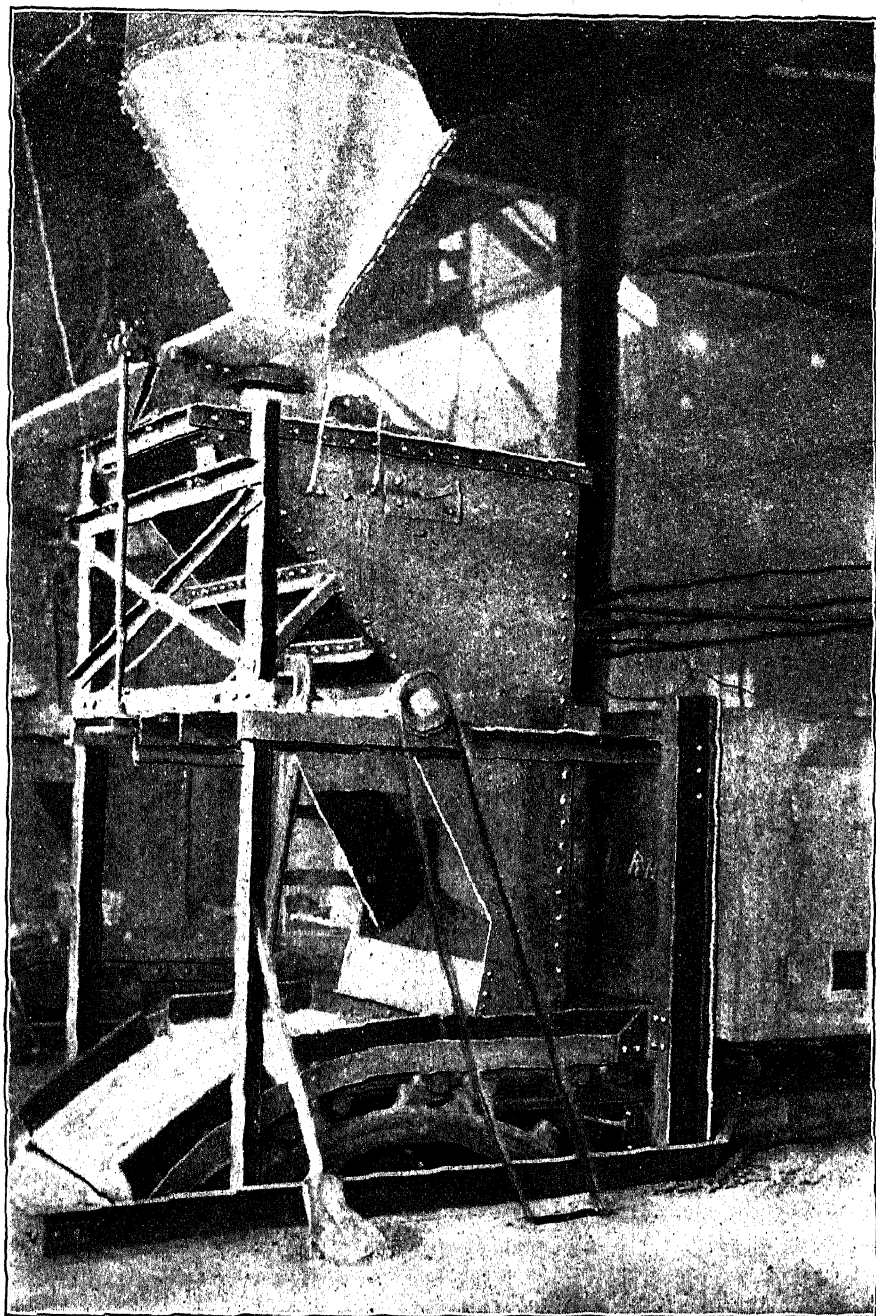
FIG. 7.—FEED DISTRIBUTOR FOR SINTERING MACHINES.

base of the crucible to the charge floor level is 30 ft., and from the center of the tuyeres to the charge floor level is 24 ft. 8 in.   All the crucibles are elliptical in shape and are constructed of heavy steel plate and angle shapes, the lower portion being imbedded in 18 in. of concrete.   The interior is lined with special fire brick and arranged with the crucible 32 in. deep and the full length and width of the furnace.   The water jackets are of heavy steel plate, all joints welded on the outside.   They are 6 ft. high, three on each side and one on each end.   Each side jacket has four tuyere openings tapered from 4.5 in. outside to 4 in. inside the face of the jacket.   The jackets are held in place by bolted angle connections and binder beams with adjustable tie rods.   Furnaces Nos. 1 and 2 have brick shafts from the top of the jackets to the feed floor.   This is supported on cast-iron columns with necessary binder rods.   Furnaces Nos. 3 and 4 have a tier of upper water jackets 9 ft. in height.   From the top of the upper jackets the furnace shafts of Nos. 3 and 4 are constructed of brick, supported on structural steel frame and columns.   These shafts are lined with cast-iron plates.   The tops of the furnaces are equipped with hinged drop doors operated by geared winches.   The blast-furnace charge is made up in the car from several different weighing hoppers which contain the predetermined amount of each material.   The charges can be weighed in the hoppers and thus be ready for immediate use on the arrival of the charge car.   The charge car is run directly over the furnace and dumped by simple release of a latch.

The gases from each furnace pass into an elevated steel dust catcher through a downtake under the charge floor.   This dust catcher is 20 ft. in diameter by 18 ft. high, with self-supporting conical roof.   The gases enter at a tangent and are drawn into the main flue through an uptake 6 ft. in diameter.   The main flue is similar in construction to the sinter-plant flue, and is about 500 ft. in length from the furnaces to the bag-house fan intake.   The dust collected in the dust catchers is withdrawn directly on to a belt conveyor system, which delivers into railway cars.

Lead is tapped into conical pots of 3 cu. ft. capacity, and trammed direct to the drossing plant.   Slag and matte are tapped either into extra large movable settlers, or into 7-ton pots to be transferred to the settling furnace.   In a case of a shut down of the settling furnace the matte is tapped from the settler into large pans of about 1.5 tons capacity.   On cooling, it is removed from the pan by a crane and subsequently shipped to the converter plant for treatment.

Settling Furnace.

This is a coal-fired reverberatory furnace 12 by 36 ft. with a grate area of 64 sq. ft. Its total capacity is 125 tons of slag and matte. The entire bottom of the furnace is inclosed in a steel plate crucible box to prevent the escape of lead.

The stack for this furnace is of self-supporting steel type, 6 ft. in diameter by 110 ft. high, and lined for 40 ft. with fire-clay brick. The flue and stack are so designed as to permit the installation of a waste-heat boiler, should this prove advisable.

Slag and matte are tapped into the furnace from pots, two openings in the roof being provided for this purpose so that two pots can be handled at the same time. Slag is skimmed from the front end into a 22-ton pot, thence to the dump. Matte is tapped at the side of the furnace into 8-ton cast steel ladles mounted on transfer cars. These cars are trammed to the converter building, where the ladles are picked up and poured direct into the converters by the converter crane.

Drossing Plant.

This is inclosed in a building 40 ft. wide, 80 ft. long, and 16 ft. high. The equipment consists of four coal-fired cast-iron kettles having a content capacity of 33 tons of lead. The kettles are surrounded by necessary steel working floors. The lead is drossed by means of a Howard press operated entirely by compressed air. The dross is returned to the charge floor of the blast-furnace plant by a mono-rail tram and elevator system. The drossing and casting operations are controlled by an electric pyrometer, which insures regularity of operation. The bullion is siphoned into cast-iron individual molds mounted on a radial supporting frame. The bars weigh approximately 100 lb. each. They are transferred to trucks, a load consisting of 30 bars. This is trammed directly over bullion scales into railway cars for direct shipment to the International Lead Refinery at East Chicago, Indiana.

Bag House.

This is of similar design and construction to the converter bag house, with the following exceptions:

There are 10 compartments with 144 bags each. The main gas and reverse current fans are of the same capacity as those at the converter bag house. Seven compartments are equipped with special cotton fabric bags and three with woolen bags. The burning floors have small cooling flues underneath, which are connected to one

One Leblanc type condenser, direct coupled to non-condensing steam turbine, for direct accommodation of steam turbine of 750-kv-a. generator set. Condenser turbine exhausts into open filter type feed-water heater.

One direct-acting jet condenser, steam driven, for the accommodation of steam engines of the D. C. generators and blast-furnace blowers.

All condenser circulating water is delivered to natural-draft type cooling towers.

Three direct-acting, duplex, tandem-compound boiler-feed pumps, outside-packed plungers. Steam cylinders exhaust into feed-water heater.

One direct-acting duplex steam fire pump.

Ashes from the boilers are handled by reverberatory furnace ash cars.

All A. C. electric power generated is distributed to the various departments of both copper and lead plants at 2,200 volts. All motor units of 75 h-p. and over use power at 2,200 volts; units of 50 h-p. and less at 440 volts; lighting power at 110 volts. Independent electric motor drives are used to advantage on belt conveyors, fans, pumps, etc.

All D. C. power generated is used for electric locomotives, sinter and blast-furnace charge cars, 50-ton tram cars, converter stands, all traveling cranes in converter plant, power house and shops; also variable speed motor drives in sintering plant, conveyor feeders, and some belt conveyors.

Steam Boiler Plant.

Steam for the power plant and general plant service is generated by the following Stirling water-tube boilers:

Four 700 boiler horse-power reverberatory waste-heat boilers.

One 700 boiler horse-power underfeed stoker fired boiler, using slack coal fuel.

Three 350 boiler horse-power hand fired boilers, using slack coal fuel.

Feed water is supplied to the boilers at a temperature of approximately 200° F. Gases from the boilers pass directly into the main reverberatory furnace flue.

MISCELLANEOUS.

Water Supply.

This is obtained from springs in Pine canyon and stored in a reservoir of about 2,500,000 gal. capacity, from which the water is con-

veyed and distributed to the various departments of the copper and lead plants by gravity pipe lines. The water is of excellent quality for both culinary and boiler-feed purposes. Scale in boiler tubes is of a soft, muddy nature and can be readily cleaned out.

The cooling water for the blast-furnace jackets is supplied from an independent natural-draft type cooling tower and is kept in circulation by means of turbine pumps direct coupled to electric motors.

Gravity supply fire tanks are located at an elevation considerably above the plants in general and connected to an independent fire line system having hydrant and hose stations with hose and car equipment distributed about the plants.

Industrial System.

The entire industrial tramming systems about the plant consist of several miles of standard 4 ft. 8.5 in. gauge tracks, with 60-lb. steel rails and substantial fastenings. All tracks on which electric locomotives and charge cars operate are equipped with overhead trolley wire system. D. C. power is used at 500 volts. The locomotives used range from 8 to 18 tons in weight, and are equipped with adequate motor power and controllers.

Laboratory, Shops, and Offices.

The laboratory consists of complete equipment necessary for all analytical and assay work required for both plants, and all control work on smelter settlements. Electricity is used exclusively for hot plates and water stills.

The shops consist of machine, boiler, blacksmith, electric, and carpenter repair shops, adequately equipped with machine tools and electric crane service to accommodate general machinery and locomotive repairs and fabrication of improved plant equipment.

Complete warehouse and office accommodations are maintained, the offices accommodating superintendent and assistants, engineering department, time-keepers, and clerical force.

METALLURGY.

Metallurgy at the copper plant presents a few novel features. Sulphur in the charge to the MacDougall roasters averages only 22 per cent. This low sulphur necessitates the use of from 1 to 2 per cent. of coal to produce a satisfactory calcine. Several of the roasters are equipped with pipe hoppers from the charge floor, which pass through the side wall and discharge on to the fifth hearth of the furnace. Siliceous ore necessary to adjust the charge to give a 41 per cent.

SiO_2 slag at the reverberatories is fed through these pipes. This ore becomes mixed with the hot calcine coming down through the furnace, as it crosses the fifth and sixth hearths before dropping into the calcine hoppers.

The charge to the reverberatories is low in copper, averaging less than 2.5 per cent. The matte produced runs about 20 per cent. of copper and is converted direct. No particular difficulties have been experienced in this operation, and two tons of ore running 70 per cent. of SiO_2 are smelted in the converter for every ton of copper produced.

At the lead plant some departures from accepted standard practice have been made. No ore-bedding system, either manual or mechanical, is in use, the material being handled as already described. No roasting equipment other than Dwight-Lloyd machines is provided. No matte crushing or roasting equipment is required.

In case of a shut down of the settling furnace the cold matte is taken to the converter plant and melted direct in the converters with about 3 per cent. of coke and 1 per cent. of coal. About 50 per cent. of the lead in the matte is recovered in the bag-house fume and 50 per cent. in the converter slag.

The sinter charge will average 16 per cent. of S, and the sinter 5 per cent. The blast-furnace slag will run from 32 to 36 per cent. of SiO_2, this being varied as the zinc percentage on the charge is high or low.

The lead plant is still in the throes of construction, and the general metallurgy has not as yet reached a state sufficiently settled to make a detailed description possible at this time.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. Casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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